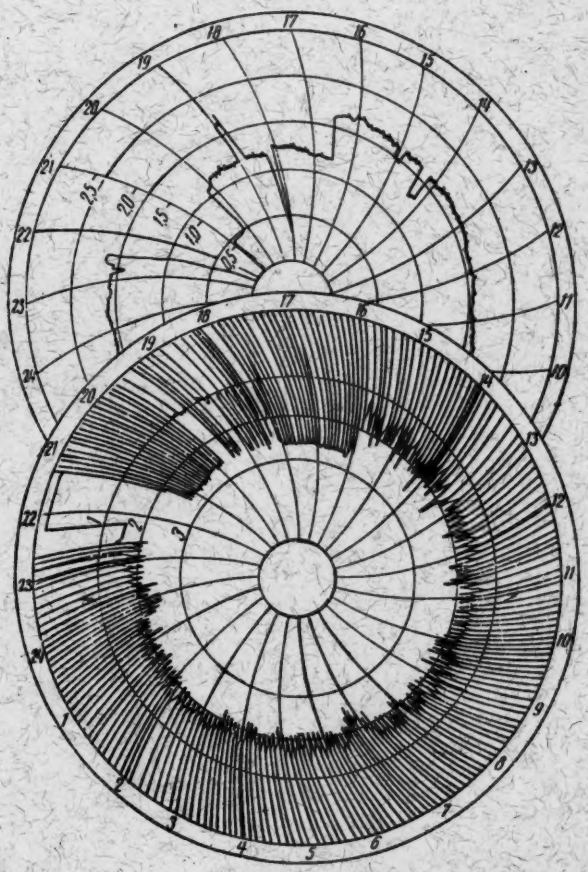
number 8

august 1958

METALLURGIST

translated from Russian



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25th ANNIVERSARY OF THE "AZOVSTAL" WORKS

G. G. Lukashov

Chief Engineer of the Works

Twenty-five years ago, on August 11, 1933, the first blast furnace of the "Azovstal" Works was blown in. That date is kept as the anniversary of the Works.

The "Azovstal" Works makes use of the enormous deposits of phosphorus-containing Kerch ores which lie almost on the surface and are transported to the Works by the Azov sea after beneficiation and sintering, and of the Donets coal processed into high-grade coke at the coke and by-product plant. Limestone required for the blast furnace and the open-hearth furnace processes is obtained from the Elenovskie quarries, situated in the vicinity of the Works. Krivoi Rog ores are also employed in the blast-furnace charge. The Works is thus situated in the center of sources of raw materials.

The constructors and the operators were quickly putting into operation one unit after another. Four blast furnaces, six 300-t tilting open-hearth furnaces, mechanical plants,and a power plant were completed by 1941. Steel-rolling plants were under construction.

The fascist aggression held up the development of the Works. The Germans did not succeed either by threats or by tortures in forcing Soviet people to carry on the production of pig iron. When retreating before the Red Army, the Hitlerites demolished the town and the Works. The returning workers found a dreadful sight: huge heaps of tens of thousands of tons of stones, lumps of concrete, mutilated steel structures and equipment were lying all over the Works.

In September, 1943, there was no water, electricity or steam either in Mariupol or at the Works.

Under those difficult conditions the metallurgists of the "Azovstal" and the constructors of the "Azovstalstroi" began the reconstruction of their Works. The seemingly impossible task was accomplished with the help of the strong will and the courageous spirit of the Soviet people.

In October, 1944, the power station was rebuilt and in July, 1945, No. 3 blast furnace was blown in. In subsequent years the reconstruction of the Works proceeded at a fast rate.

At present, the sinter plant, four blast furnaces, twelve large-capacity tilting open-hearth furnaces, the blooming mill, the rail-structural and the heavy-section mills, the plant for rail-joint bars, a slag-processing plant and other plants are in operation. Work is about to be completed on the construction of No.5 blast furnace, which will be operated at 1.5 atm top pressure. Zhdanov, the town of metallurgists, has risen from ruins and expanded considerably at the same time as the Works was rebuilt.

The Works has an efficient production presonnel. With persevering efforts and continuous political and technical education the builders and the personnel of the Works solved many a difficult production problem. The output of pig iron is now 40% greater than in 1950, coke consumption has fallen by 20% and the dust carry-out has dropped from 264 kg to 49 kg per ton of pig iron.

At present, the "Azovstal" Works,in cooperation with the Azov and the Black Seas Transport and the Kamysh-Burun Iron Ore Combine,is making arrangements for the delivery of hot,uncooled, fluxed sinter to the blast furnaces. The lighter "Mga" of 3500 t is now being refitted at the Kerch Ship Repair Yard for this purpose. After the "Mga" the whole ore transport fleet will be refitted.

Steel is made at our Works in tilting open-hearth furnaces in accordance with a special procedure, developed at the Works, which involves the retaining of some steel in the furnace after the removal of slag and part of the metal. Phosphorus-containing pig iron (up to 1.5% P) can be efficiently converted into steel by this method. In the course of the process the slag is repeatedly run off and new slag is formed. Primary slags are processed in the slag processing plant into valuable phosphorus-containing fertilizers of which several hundred thousand tons are supplied to the collective farms annually.

Oxygen is extensively used in the open-hearth process; it is used in the flame and in the bath. All furnaces are provided with magnesite-chromite roofs. A year ago the builders handed over to the operators the last open-hearth furnace and it has been successfully put into operation.

This year the rolling mill operators of the Works mark the completion of the first ten years of the operation of the steel rolling plants. Continually adopting advanced practices of leading metallurgical establishments, the operators are not inferior to rolling mill operators at the best steel works in the country in respect to the productivity in a busy time or the quality of production.

The removal of slag in a liquid state from the soaking pits of the blooming mill; the mastering of the rolling of 25-m rails; the rolling of rails in seven passes according to a new pass design, instead of ten; the insulation of the gliding tubes of the holding furnaces; the plating, hardening and ragging of rolls; the improvement of the equipment in the rail-joint plant; the mastering of the production of new type rail-joint bars – this is far from a complete list of the achievements of our rolling mill operators.

The staff of mechanics and power engineers endeavors to improve equipment components and to develop more efficient methods of maintenance and repair of equipment. Extensive projects on the automation of metallurgical processes, the application of atomic energy for industrial purposes, the rise of operating efficiency, an improvement in the quality of the product, and a reduction of production costs are under way at the Works.

Although the Works has developed large-scale production and comprises a complete cycle of iron and steel industrial processes, there is still great scope for expansion. The erection of No. 5 and No. 6 large-capacity blast furnaces at the Works and the building of No. 5 and No. 6 sinter machines at the Kamysh-Burun Combine (the main iron-ore base of our Works) make the erection of a second steelmaking plant and a second group of rolling mills very urgent. These constructions will cost us much less than the construction of the initial set of plants, as a lot of the foundation work is already done. At the same time, the inadequacy of the repair services and the shortage of the machine stock is felt very strongly. An improvement in this direction is imperative. The operation of open-hearth furnaces is made difficult as there is only one mixer for the whole open-hearth plant; usually there are two mixers at plants of similar (or even smaller) output. In order to increase the steel output in 1959, it is necessary to install a second mixer.

In view of the high speed of steel rolling, it is necessary to erect a fourth holding furnace in the rail-structural plant.

Extensive building of dwelling houses is envisaged for the next few years. The completion of the accommodations provided for in the plan should fully meet the needs of metallurgists in housing. To solve the housing problem we need effective assistance from the Stalino Sovnarkhoz.

Following the great industrial traditions developed at the Works, adopting the experience of leading workers, and making use of new potentialities of output increase, the workers of "Azovstal" fight successfully for the fulfillment of the 1958 production plan ahead of schedule.

BLAST FURNACE PRODUCTION

COMPACTING OF IRON ORES AND CONCENTRATES

N. A. Iarkho

The compacting of iron ores and concentrates by means of nodulizing and subsequent fusion has been known for over 40 years. However, only in the past decade has this method begun to be employed industrially in Europe (Sweden, Norway, Finland) and in particular in America (USA and Canada).

The method of nodulizing was developed somewhat later than sintering, which began to be employed at the end of the past century. While, however, sintering became an integral part of blast furnace industry, nodulizing was not used for almost thirty years.

As the deposits of rich lumpy ores became depleted, it was necessary to make use of lean ores and ferrous rocks which could not be employed in the blast furnace process without a preliminary beneficiation. The products in the benefication of such ores are fine concentrates whose main bulk constitutes particles of under 0.1 mm size; hence, they have to be compacted before being charged into the blast furnace.

Transportation of nonbeneficiated ore is uneconomical; therefore, the beneficiation should take place where the ore is mined. The transportation, however, of fine concentrates entails large losses due to dust formation, and in wintertime unloading of railroad cars becomes difficult because the concentrates freeze. It is difficult to sinter fine concentrates because of poor gas permeability. The vertical velocity of sintering constitutes only 10-12 mm/min and the output of a sintering plant is 0.5-0.8 t/sq m hr. Moreover, if the sinter is produced at the mine, the particles will get reduced in size during the transportation, and the compacting process will be brought to nought.

All these factors brought about the development of the method of nodulizing which makes it possible to overcome the above difficulties.

Sample Screen Analysis of Ore Fines for Sintering and Nodulizing

Size of openings in the screen, mm	Residue on the screen, %	
	ore for sintering	ore for nodulizing
10	4,1	—
8,0	2,3	—
3,0	11,7	—
1,0	7,0	—
0,5	27,1	—
0,3	18,2	—
0,15	20,3	0,4
0,09	2,1	11,6
0,075	4,1	—
0,06	3.1	11,3
Below 0,06	—	76,7

Sintering and nodulizing are frequently regarded as two alternative methods, but this is not so. The table on page 393 shows clearly the different ranges of application of sintering and nodulizing, depending on the particle size of the initial raw materials.

The screen analyses of ore intended for sintering and that intended for nodulizing show that sintering and nodulizing do not exclude each other but are complementary. Ore fines which have a high content of relatively large fractions, and are produced as a result of ore crushing and screening, are subjected to sintering at the metallurgical works. Fine ore concentrates are obtained as a result of the beneficiation of lean ores and ferriferous rocks at the beneficiation plants near the mines and they should be nodulized there.

Fused nodules have some advantages over the sinter.

1. The nodules are of spherical shape, more porous, and of quite uniform size. These factors ensure a good gas permeability of the charge in the blast furnace.

2. Owing to the high degree of oxidation achieved as a result of roasting moist nodules, the reducibility of the nodules is approximately the same or even higher than the reducibility of hematite ore and considerably higher than the reducibility of the sinter.

3. The nodules have a high mechanical strength which is retained on heating, while the mechanical strength of ore drops by half at 800° C.

4. The nodules are only slightly crushed on bulk loading and several transfers. They can be kept in piles in the open for a long time in any weather.

Mining in the USA of taconite (lean ferruginous rock distinguished by fine dissemination of useful minerals) deposits which produce, after beneficiation, ore concentrates containing particles of up to 0.04 mm size, was conducive to the study of nodulizing and the investigation of its industrial application. After the first experiments, large commercial experimental plants (Aurora, Babbit and others) of 200,000-300,000 t annual output of nodules were built. The next step was the erection of large scale commercial plants of 3,600,000 t and 7,600,000 t annual output of fused nodules in Minnesota by the Reserve Mining Co. and the Erie Mining Co. respectively.

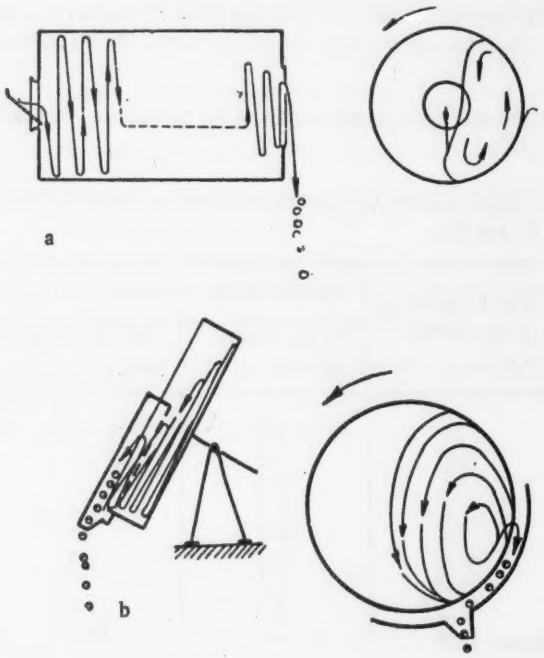

Fig. 1. Diagrammatic representation of nodule formation:
a) In the drum; b) on the plate granulator.

The first experimental plant in Europe of 7 t per 24 hrs output was built in Sweden (Sandwicken) in 1946 - 1947. Subsequently, experimental plants were built in England, Norway and the Federal German Republic.

In 1955, there were already seven commercial plants in Sweden with a total annual output of 250,000 t. The nodules produced are used mainly for direct iron production by the Viberg method and in the steelmaking industry. A commercial plant for nodulizing and roasting of pyrite cinders with the removal of sulfur and side recovery of copper zinc and lead is in operation in Imatra (Finland). The plant processes 120-130 t of pyrite cinder in 24 hrs.

The process of nodulizing consists of producing moist nodules which are subsequently hardened by roasting. Moist nodules are produced by mixing finely ground ore concentrate with the necessary amount of water and treating the mass in special rotary devices. The nodules are formed as a result of the interaction between the ore concentrate particles and the water. The extent of the interaction depends on the properties of the material, its wettability, and the size and shape of the particles. The smaller the particles of the concentrate and the smaller the voids between them, the stronger are the nodules produced. In the process of nodulizing, water presses out the air from the voids between the particles of the concentrate and, as it were, draws them together.

Rotary drums or plate granulators are employed for the nodulizing operation. The rotary drum is the predominantly used machine in industry plate granulators are employed less frequently. Vibration effect can also be made use of for the nodulizing process. However, owing to the complexity of the equipment required, this method has not yet been introduced on a commercial scale.

In drums and in plate granulators, the nodules are formed from concentrate particles in a similar manner to snowball formation. The diagram of the movement of the particles of the charge in the rotary drum is shown in Fig. 1.

The output of the drum constitutes 120-1000 t/24 hrs and it depends on its dimension, inclination, peripheral speed, and the percentage of drum volume occupied by the charge. In the USA, at the Davies Works in Silver Bay, rotary drums of 2.74 m diameter, 9.15 m long, lined with concrete and rotating at 30-40 rpm are

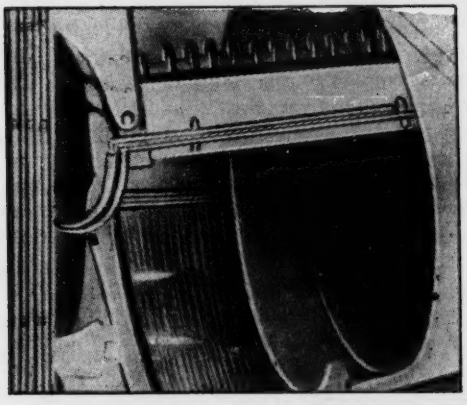

Fig. 2. Rotary drum with the discharge part in the form of a screen.

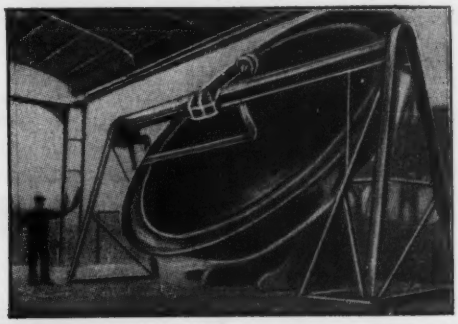

Fig. 3. 5 m diameter table granulator with a ring trough.

in operation. Their output is 30-40 t/hr. With an increased speed of the drum the output increases. But at too high a speed of rotation, the conditions become less favorable for nodule formation and the nodules formed get reduced in size. At the discharge end of the drum, screens are installed for the separation of fine fractions which are then returned to the drum. Sometimes the drum discharge end itself is made in the form of a screen (Fig. 2). The drum is usually inclined at 2-6°. With an increase in inclination the yield of fine fractions increases. On the other hand, a decrease in the angle of inclination results in an excessive increase in the size of nodules and a wider range of sizes.

Plate granulators (Fig. 3) can produce nodules of a fairly uniform size. As a result of segration in the course of nodulizing, large particles accumulate at the rim and smaller ones near the center of the granulator (Fig. 4). In this way, only the nodules of predetermined sizes are discharged. Hence, a separate operation for the size separation of nodulized material can be dispensed with.

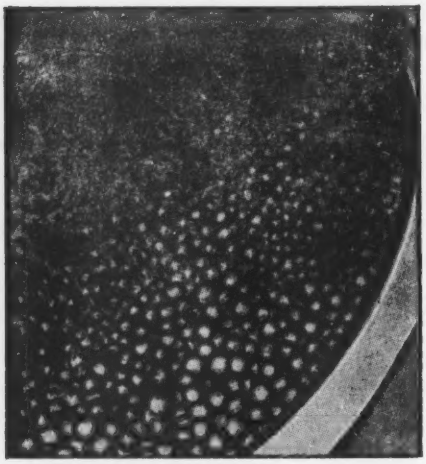

Fig. 4. The distribution of nodules near the rim of a plate granulator.

At present, granulators of 3.6-5.5 m diameter, operated at 30-90 m/min peripheral velocity and having an output of 3-25 t/sq m per 24 hrs, depending on the type of raw materials, are employed. The nodules in the granulator can be covered with a layer of another material. For this purpose, a shallow circular trough is fitted outside the rim of the granulator cup. The nodules, when discharged, fall over the brim of the granulator into the trough and, rolling in the trough, cover themselves with the material supplied. This is an important advantage of the granulator over the rotary drum.

A method of nodulizing iron ore concentrates of large particle size, involving a preliminary compression of the material, was developed in Germany. The ore concentrate was compressed on vacuum belt presses and the cylindrical pellets were rolled in a drum and roasted. This method allows the use of initial material of large particle size and the production of uniform size pellets.

A similar method was developed in the Soviet Union at the Moscow Steel Institute. An experimental press designed on this principle has been installed at the Dzerzhinskii Works.

For a normal operation of a belt press a definite plasticity of the ore is necessary, and this is sometimes achieved by the introduction of special additives. The disadvantage of this method is the limited range of permissible moisture content in the charge. If the moisture content is too low, the material loses its plasticity, gets overcompressed, and the press stalls; if the moisture content is too high, the rolling of the pellets in the drum becomes difficult and the physical properties of moist nodules deteriorate.

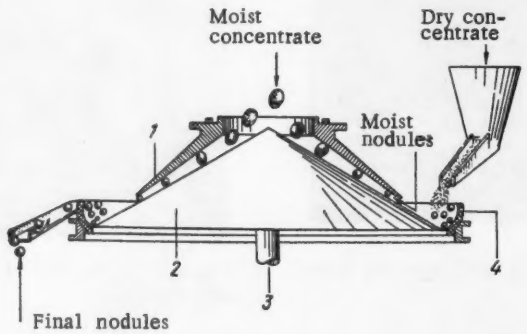

Fig. 5. Diagram of the cone granulator: 1) stationary cone; 2) rotary cone; 3) driving shaft; 4) rubber rim.

These difficulties are eliminated by a special granulation (Fig. 5), developed recently in Canada, for rolling cylindrical pellets. Ore concentrates, containing up to 15-25% moisture can be preliminarily pressed if this granulator is employed. Cylindrical pellets containing 15-25% moisture constitute the charge for the granulator. Dry ore concentrate is also fed in and is rolled onto the pellets. In this way, nodules of uniform size and with determined moisture content can be produced.

The moist nodules must be strong,to withstand transport to the roasting kiln and not get crushed or deformed, and they must be thermally stable, i.e. capable of withstanding rapid heating without disintegration. Various additives which enhance the mechanical strength and thermal stability of the nodules are employed. When the additives are introduced, thermal shock temperature (at this temperature, cracks appear on moist nodules owing to rapid heating) increases from 200 to 600° or more. Most of the additives in use exhibit a substantial effect on the mechanical properties of roasted nodules and increase their strength and their resistance to attrition. The most widely used additives are bentonite and various salts ($CaCl_2$, NaCl, sylvinite and others). These materials are usually added in small quantities (0.25-0.5%) to the charge.

The nodules are subjected to roasting at 1150-1350° C. The choice of roasting temperature depends on the properties of the concentrate, the composition of the nonmetalliferous rock, and the kind of additives used. When the nodules from magnetite concentrate are roasted, the hardening is achieved on account of the recrystallization and the growth of hematite crystals formed as a result of oxidation. This process starts at the temperature of about 200-300° C. After the roasting, hard oxidated nodules consisting almost entirely of Fe_2O_3 are obtained. Their porosity amounts to 20-25% and their weight is about 2.2 t/cu m. When the nodules of hematite concentrate are roasted, the strengthening takes place at a temperature approaching the softening temperature of the concentrate, and correspondingly more heat is required for roasting.

Until recently, the shaft kiln of rectangular or circular cross section was the most widely employed furnace in industry for roasting nodules. A uniform charging of moist nodules can be more easily effected in the rectangular furnace and thus a more uniform distribution of gases across the furnace can be ensured. The output of the furnace depends on the size of the nodules,and it reaches 800 t/24 hrs. Better characteristics of shaft furnace operation have been attained on roasting magnetite nodules, as in this case heat is produced inside the nodule on account of the oxidation of magnetite. When nodules of hematite concentrate are roasted all heat required has to be supplied from outside, and the temperature of roasting is near the softening temperature, therefore, it is rather difficult to ensure a uniform temperature across the furnace and to avoid local overheating and the fusion of nodules into large lumps.

American practice is to introduce sometimes, in addition to binding material, fuel material in the form of finely ground anthracite or coke into the nodules. The total amount of added fuel material does not exceed 1-1.5% for magnetite nodules and about twice as much for hematite nodules. The addition of fuel is more typical of American practice; in Sweden it is rather an exception. Added fuel material is either rolled onto the surface of the nodules or is mixed with the initial concentrate and is evenly distributed in the nodule.

Recently the shaft furnace is beginning to be replaced by a more efficient equipment for roasting – a chain machine similar to the sintering machine.

At the Copper Cliff Works in Canada a machine of 1100 t/24 hrs output, made by the Lurgi company (West Germany) is in operation. The nodules, without the addition of a binding agent or fuel, are fed into the machine over a width of 2.5 m in a layer about 500 mm thick. Combustion products of fuel oil, burnt in a special hearth,penetrate through the layer. The sides and the grates of the plates are protected from hot gases by a special bed layer. When the hot gases are drawn through downwards, the nodules are dried and roasted and when the cold air is drawn upwards the cooling of the nodules takes place. Hot flue gases from the final vacuum chambers and the air from the rear part of the machine are taken to the hearth.

At the Silver Bay works (USA) chain machines 51.2 m long and 1.83 m wide, equipped with 28 vacuum chambers, are used for nodulizing. The hearth is made in the form of a furnace consisting of three sections: drying, preliminary heating,and roasting sections. Six fans, instead of the exhauster, are installed. Longitudinally the machine can be divided into five main zones. The first seven vacuum chambers, operating with the draft upwards, are provided for the preliminary drying of moist nodules. The moisture content of the nodules is reduced from 10% to 5%. Above the next seven vacuum chambers, operating with downdraft, the three-section kiln is situated. Next follows the recuperation zone where the entering air is heated by the leaving gases, and then come the two cooling zones.

At the Eagle plant (USA), machines of 2000 t/24 hrs output and having a chain 1.83 m wide and 68 m long are in operation. The nodules are roasted along the section 51 m long and are then cooled. On the 100 mm-high bed a thin layer of anthracite is spread and it is ignited under the hearth. Over the first 5 vacuum chambers the bed layer is brought to red heat to the depth of 50 mm. On the red hot bed the first layer of nodules, rolled

in anthracite is charged to the height of 200 mm. By that time the plates pass above the vacuum chambers which provide upward draft and the center of combustion is moving upwards. When the temperature of flue gases reaches a given temperature, the next layer of moist nodules is charged. In this way the charged layer is brought to 915 mm height.

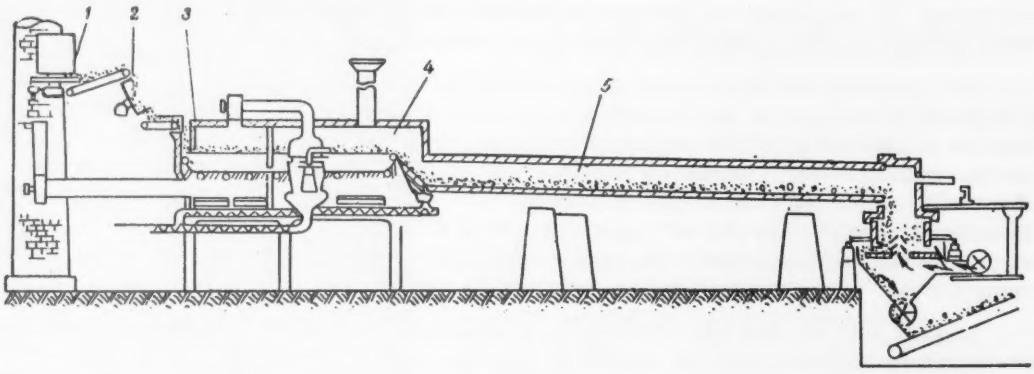

Fig. 6. Plant with the rotary kiln for nodulizing: 1) bin for the feed; 2) plate granulator; 3) drying chamber; 4) chamber for preliminary heating; 5) rotary kiln.

An economical method of roasting moist nodules in a rotary kiln (Fig. 6) has been developed in the USA. Moist nodules are fed by a conveyor onto a moving grating which passes through a furnace divided into two compartments: a drying chamber and a chamber for preliminary heating. The gases, leaving the preliminary heating chamber at a temperature of 300-400° C, are drawn through the layer of nodules in the drying chamber. Dried nodules enter the heating chamber and there they are subjected to the action of hot gases at 1000-1100° C of high oxidizing power from the rotary kiln. As a result of the oxidation of magnetite, the nodules harden and easily withstand subsequent roasting in the rotary kiln without being crushed. The maximum strength of the nodules is attained by roasting them in the rotary kiln at 1300-1350°. In the discharge compartment of the furnace, the nodules are cooled with the air which thus gets warmed before entering the furnace proper. The successive utilization of the countercurrent principle with regard to the gases and the nodules in all stages of the process (from the drying to the cooling of the nodules) means a considerable economy in fuel.

Strong particles containing, in addition to iron ore concentrate, carbon and flux can be obtained without roasting. A catalytical chemical method, developed in the USSR, of producing iron ore and coal pellets is based on the conversion of slaked lime, added to the pellets, into $CaCO_3$. By treating moist pellets with flue gases which contain carbon dioxide, it is possible to produce strong water-resistant pellets. The above process proceeds successfully only at a definite moisture content (about 5%) and in the presence of a catalyst.

Quite different from the above commonly used methods of roasting nodules with the object of strengthening them is the method of producing nodules and a simultaneous reduction of iron oxides to metallic state in the process of roasting. Relevant experiments are being carried out at the TsNIIChM (Central Scientific Research Institute of Ferrous Metallurgy) in the Soviet Union. Nodules containing a large amount of a reducing agent (anthracite) are subjected to roasting at 1250-1300° C. Strong nodules, in which up to 80% of iron is reduced to metallic state, are obtained. Foreign technical literature recently reported similar experimentation in the USA, where it is intended to use partly reduced nodules as charge for electric blast furnaces. According to preliminary calculations, the output of the electric blast furnace can be raised to 400 t/24 hrs by the use of such a charge, and at the same time electric power consumption can be reduced to about half the value required for the nonreduced charge. Also, a method of direct reduction, based on the treatment of ore and coal lumps on the sintering machine, has been proposed in the USA. It is expected that in the process of roasting strong lumps, containing up to 40% of iron oxides reduced to metallic state and about 6% of residual carbon, will be obtained.

To estimate the quality of final fused nodules, the fall and the attrition tests have been developed. In the

latter, a 22.7 kg sample of nodules is charged into a 610 mm diameter drum containing two baffles. After 200 revolutions, the amount of fines (particles below 0.5 mm) should not exceed 12%. This is about three times the attrition met with on the transportation of nodules by ships.

The compacting of fine concentrate is of great importance for our country. The exploitation, envisaged for the near future, of enormous deposits of ores requiring extensive beneficiation, which entails the fine grinding of raw materials, will call for an extensive application of advanced methods of compacting, which would ensure the production of strong, lumpy, and transportable material, suitable for use in the charge of modern large blast furnaces. One of these methods is undoubtedly nodulizing combined with subsequent roasting.

PRODUCTION OF CONVERSION PIG IRON WITH THE COMPLETE ELIMINATION OF MANGANESE ORE FROM THE CHARGE

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Stalino Metallurgical Works

The production of low-manganese conversion pig iron permits an increase in the output of blast furnaces and a considerable reduction in production costs without capital expenditure.

A prolonged production of conversion pig iron of low manganese content (0.5-1.0%) was effected for the first time in our country in 1952, at the Magnitogorsk Metallurgical Combine; afterwards the Combine began the production of conversion pig iron containing 0.2% manganese.

It is difficult to produce that type of pig iron in the South owing to the high sulfur content in coke and high coke consumption. Nevertheless, several works in the South have mastered the production of low-manganese pig iron, containing 0.8-1.2% manganese. At our Works, such a pig iron was produced from October, 1955, to September, 1957, the consumption of manganese ore being reduced by 50-70%. The output of the blast furnaces increased by 6%, coke consumption fell by 6%, and the cost of 1 ton of pig iron was reduced by 15-20 rubles.

In September, 1957, for the first time in the South, the production of low-manganese pig iron (0.22% Mn) began in No. 3 blast furnace, manganese ore being completely eliminated from the charge.

When manganese ore was completely eliminated, the performance of the furnace improved still more (see Table).

Recalculated on the basis of production without metallic additions, the output of the blast furnace increased by 3.4% and coke consumption fell by 2.1%.

The elimination of manganese ore was, at first, accompanied by an increased consumption of dolomitized limestone as the intention was to obtain 4.5-5.0% magnesia content in the slag. However, continued operation of the furnace showed that magnesia content in the slag can be decreased to 3.0-3.5% at 1.28-1.30 slag basicity and not less than 7.0% (within the limits of 7.0 and 10.0%) alumina content.

The increase of slag basicity ($CaO : SiO_2$) to 1.28-1.30 allows a rise in the temperature of the blast, since the level of slag formation is lowered owing to the absence of easily fusible manganese silicates, and the gas permeability of the furnace stock improves, the improved permeability being enhanced also by the elimination of the dust-bearing manganese ore.

Composition of iron ore portion of the charge, %

High-alumina ore from the Dzerzhinskii Ore District	51.4
Ore from the Ingulets Ore District	24.5
Magnetite ore, class 20	17.9
Quartzite ore, class 25	3.8
Clinker	2.4

The average lumpiness of the ores was 75-80%. Ore was filled in cycles, each cycle consisting of five charges, for a more uniform distribution across the shaft of alumina-containing ores. The basic manner of filling the furnace was three charges ore-ore-coke-coke-coke-stone and one charge coke-ore-ore-coke-coke-stone, the weight of coke charge being 4350 kg. Normal level of the stocking was 1.5 m.

Characteristics of the Blast Furnace Operation after the Changeover to the Production of Pig Iron without Manganese Ore

Characteristics	Period of operation		
	Aug. 1956, to Aug. 1957. Partial elimination of manganese ore	Sept., 1957. Transitional period	Oct., to Dec., 1957. Complete elimination of manganese ore
Coefficient of furn. vol. utiliz.	0,904	0,895	0,892
Pig iron output without taking into account metallic additions, %	100,0	103,2	103,4
Pig iron composition, %			
Si	0,70	0,87	0,88
Mn	1,06	0,52	0,22
S	0,035	0,033	0,032
Slag basicity ($CaO : SiO_2$)	1,28	1,27	1,29
Materials (t/t of pig iron):			
iron ore	1,777	1,825	1,800
manganese ore	0,077	0,026	—
limestone	0,719	0,804	0,791
dolomitized limestone	0,276	0,298	0,265
coke (moist)	1,043	1,052	1,070
Dust carry-out (t/t pig iron)	0,118	1,155	0,137
Slag (t/t of pig iron)	1,053	1,150	1,090
Intensity of process t/cu m 24 hrs	1,15	1,17	1,20
Iron ore portion of the charge, %	53,3	52,5	52,8
Coke consumption on the basis of pig iron production without metallic additions, reduced to slag yield of 1,053 (t/t) pig iron	1,104	1,024	1,081
Ditto, %	100,0	93,0	97,9

Carbon dioxide content was 6.0-7.0% at the periphery of the stock and 4.0-5.0% at the center. If the center of the stock was overburdened with the ore charge the filling of the furnace was changed to the coke-

ore-stone-ore-coke-coke-stone system and if the periphery was overburdened the coke-ore-ore-coke-coke-stone system of filling was applied.

Hence, for the production of low-manganese pig iron under the conditions obtained at the Stalino Works, it is necessary:

1) to change the slag regime and to introduce about 60% of alumina-containing ores from the Dzerzhinskii District into the charge;

2) to bring the slag basicity ($CaO : SiO_2$) to 1.28-1.30;

3) not to reduce alumina content in the slag below 7.0-10.0%, and to increase MgO content to 3.0-4.0% by introducing 20-25% dolomitized limestone into the charge.

4) to somewhat increase the temperature of the hearth (silicon content in pig iron should constitute 0.7-0.9% and the temperature of pig iron should be 1470-1500° C);

5) to employ high-temperature blast (temperature of the blast should be 750-800° C).

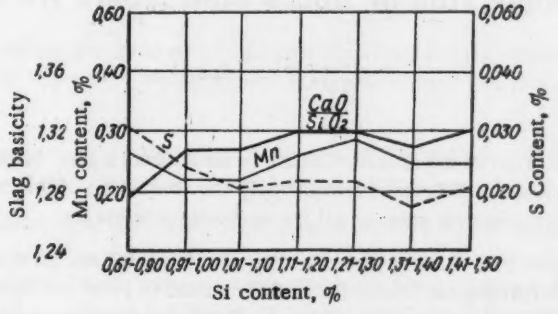

Fig. 1. The effects of the intensity of hearth heating and slag basicity on sulfur content in pig iron.

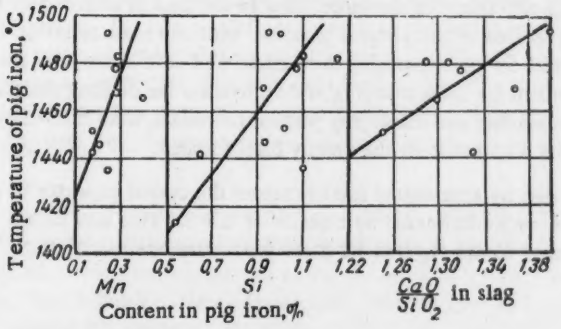

Fig. 2. Relationship between the composition of pig iron, the basicity of slag, and the temperature of pig iron.

The quality of pig iron with respect to sulfur content improved markedly. Fig. 1 represents the data on the quality of pig iron during the period from September 24 - September 30, 1957. It is seen from the graph that with the increase in the heating of the hearth and with the increase in slag basicity, sulfur content in pig iron decreases. The temperature of low-manganese pig iron is higher than that of ordinary pig iron. It is 1440-1490°C, an increase in temperature (Fig. 2) and in the fluidity of pig iron being observed with the increase in silicon content in pig iron and the increase in slag basicity. The losses of pig iron at the casting yard were

reduced markedly, working conditions at the pig iron runners improved, and the output of cold pig iron increased.

For an extensive introduction of low-manganese pig iron at the works in the South, more regular supplies of iron ore and dolomitized limestone have to be planned.

Editor's note. Experience from the operation of the Dzerzhynskii Works indicates that it is possible to obtain residual slag of entirely satisfactory physical and chemical characteristics even at a lower (5.0-5.5%) alumina content than reported by the authors. It is then necessary to have 5.5-6.0% MgO in the slag, the basicity $CaO : SiO_2$ should be 1.28-1.30, and the ratio $(CaO + MgO + MnO) : SiO_2 = 1.45 - 1.46$.

PRODUCTION OF MOLDS FROM LIQUID PIG IRON

G. I. Shandrenko

The information given in the article "Casting of molds from liquid blast furnace metal" by P. V. Gubchevskii (Metallurgist, No. 1, 1958) shows convincingly the advantages of the use of liquid conversion pig iron for mold casting, but no account is given of all the economic advantages.

Referring to the practice of many years standing of the specialized plant for mold casting from liquid pig iron at the Krivoi Rog Metallurgical Works, the author omitted to point out that the surplus floor area in that plant offers the possibility of increasing the output far above the design capacity of 100,000 tons per year.

This statement is based on the fact that the plant at the Edgar Thomson Works of the United States Steel Corporation in Braddock (Pennsylvania, USA) built in 1952, and designed for the production of 250,000 tons of molds per year, has 8100 sq m floor area (Foundry, N.V., 1955, III, v. 83, No. 3, p. 100-105), and the mold plant at the Krivoi Rog Metallurgical Works has 10,400 sq m floor area.

The high operating efficiency of the mold plant in the USA is attained by the mechanization of production processes. The molding section of the plant is provided with two units installed in parallel and each consisting of a sand-throwing machine for sand-ramming the pattern and core boxes, two revolving tables of 7.6 m diameter, four drying ovens, equipment for flask stripping, and a chamber for cooling the flasks and the cores. All the operations on loading, unloading and conveying pattern materials, used sand clay, and other materials are mechanized. Liquid pig iron is supplied from the nearby blast furnace.

Hence, it follows that by appropriate modifications the output capacity of the mold plant at the Krivoi Rog Metallurgical Works can be increased by a factor of 2.5-3. This fact should be taken into account when the economics of the use of liquid pig iron for mold production are calculated.

THE STEELMAKING INDUSTRY

PRELIMINARY TREATMENT OF PIG IRON WITH STEAM-OXYGEN MIXTURE IN THE LADLE

Ia. A. Shnerov, A. G. Derfel, A. G. Kotin, M. T. Bul'skii and A. G. Alimov

The Ukrainian Institute of Metals and the "Azovstal" Works

With the object of extending the field of applications of oxygen and its economic utilization, the Ukrainian Institute of Metals and the "Azovstal" Works carried out experiments on the preliminary treatment in ladles of pig iron with oxygen.*

The work carried out constitutes the first stage of the research program which envisages the carrying out of heats in open-hearth furnaces with the use of oxygen-treated pig iron.

According to the published data, the content of silicon in pig iron falls from 0.7 to 0.3-0.4% after the preliminary treatment of pig iron, and the output of open-hearth furnaces increases by about 15%. The higher the content of pig iron in the open-hearth furnace charge, the more advantageous is the preliminary treatment of pig iron. This, in particular, applies to the "Azovstal" Works where pig iron constitutes 75% of the metal part of the charge. Furthermore, the lowering of silicon content on treatment of phosphorus-containing pig iron will make it possible to produce phosphate slags of a higher P_2O_5 content.

For the study of the regime of preliminary treatment of pig iron in ladles, a semi-industrial plant (Fig. 1) was built in the mixer building at the Azovstal Works.

130 ladles, including 114 ladles with phosphorus-containing pig iron and 16 ladles with ordinary open-hearth pig iron, were blown through. Pig iron was blown through in the ladles from the blast furnace plant through a vertical, 1-inch-diameter iron tube which was immersed 150-250 mm below the slag level. The tube was protected to the height of 1800-2100 mm with refractories. The first nine ladles were treated with pure oxygen, and in the later experiments a steam-oxygen mixture (20-40% steam content by weight) was employed for blowing through pig iron. At the above ratio of steam to oxygen it was possible to avoid the evolution of brown fumes.

Effect of additives. In addition to the tests on pig iron without additions, the following different cases of additives were investigated in the treatment of phosphorus-containing pig iron (% by weight of pig iron): limestone – 1.5% and 3% (in the latter case 1% ore was added for thinning); ore – 2.5% and 5%; ore and limestone – 1.5% and 2.5% each. The following cases of phosphorus-free pig iron with additions were studied: limestone – 1.5%; ore – 1.5%; ore and limestone – 1.5% each.

The analysis of the data on the treatment of phosphorus-containing pig iron showed that in the course of filling and transporting the ladles which contained no additives, the content of silicon remained unchanged, manganese and carbon contents decreased by 0.04-0.05% (about 2% of initial content) and sulfur content decreased by 0.027% (about 25% of initial content). The additives affect the content of inclusions – mainly silicon, manganese and sulfur – substantially. The largest reduction in the content of Si and, in particular, Mn was

*B. S. Kurapin, V. E. Koval', V. I. Khmirov and N. T. Berilov took part in the work.

observed in the cases where one alone was added (2.5% and 5%); it constituted on the average 0.22% and 0.34% Si (31% and 60% of the initial content) and 0.55% and 0.94% Mn (30% and 58% of initial content) respectively, whereas the content of sulfur in the samples not only showed no decrease but even increased by 0.005 and 0.015% respectively.

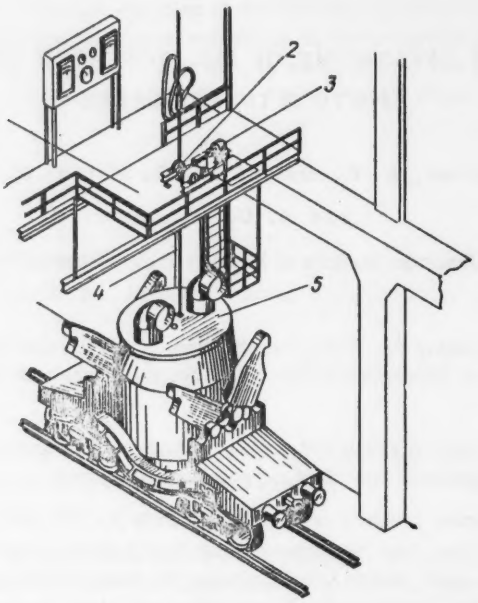

Fig. 1. Diagram of the equipment for blowing-through pig iron in the ladle: 1) cross bar; 2) hoist; 3) electric motor for the hoist; 4) blowing tube; 5) cast steel lid

In the case of the addition of limestone alone (1.5%), and limestone (3%) with a small amount of ore (1%), the silicon content decreased during the filling and the transportation of the ladle by only 0.02% and 0.08% respectively (4% and 8% of the initial content), and manganese content decreased by 0.13% and 0.24% (7% and 14% of the initial content). At the same time the sulfur content decreased by 0.048% and 0.063% (40% and 52% of initial content) – more markedly than in the case without additives.

In the two cases of mixed additions of limestone and ore (1.5% each and 2.5% each) the silicon content decreased by 0.12% and 0.11% (18% and 17% of the initial content) and manganese content decreased by 0.29% and 0.38% (16% and 18% of the initial content), i.e., less than in the cases with the addition of ore only and more than in the cases with the addition of limestone only. The two cases of mixed additions occupied an intermediate position also with respect to the change in sulfur content which decreased by 0.040% and 0.037% respectively (32 % and 30% of the initial content).

The additives show a marked effect also during the process of blowing pig iron. When pig iron without any addition was blown through, 0.16% Si on the average was removed in the course of 30 min; when pig iron with various additives was blown through, on the average 0.25% to 0.31% Si was removed, i.e., 1.5 to 2 times more. The same action of the additives was also observed after 45 min blowing: 0.20% Si and 0.30% – 0.36% Si was removed from pig iron without and with additives respectively. The amount of silicon burnt out from pig iron with the addition of ore alone or ore and limestone was approximately the same as from pig iron to which limestone alone was added. Ore and mixed additions have no effect on the burning out of manganese during the blast; in the case of the addition of limestone alone, or lime, the amount of burnt-out manganese decreases by a factor 1.5.

Sulfur content changes only a little in 30 min and increases slightly in the subsequent 15 min.

As a rule, the sulfur content in pig iron after the blast is lower by 0.02%-0.05% than at the tapping; the larger the decrease in sulfur content during the tapping and transport the lower is its content after the blowing (cases of the addition of limestone alone or limestone and a small amount of ore). Phosphorus content in pig iron did not increase either during the process of filling the ladles with pig iron or in transportation, nor in the process of blowing.

The burning out of carbon during the blast is relatively small (0.2-0.3% in 30 min and 0.3-0.4% in 45 min); no definite dependence on the type of additions introduced into the ladle during filling it with pig iron was detected.

The blowing of phosphorus-containing pig iron for more than 30 min is unadvisable since the silicon content decreases very little in the course of the next 15 min and the sulfur content even increases.

The effect of the additions on the removal of the inclusions during the filling of ladles at the blast furnace plant, and during ladle transportation and blowing, in the case of phosphorus-free pig iron is similar to the effect observed in the case of phosphorus-containing pig iron.

In the course of the process of blowing, in connection with the oxidation of Mn and Si, the content of MnO and SiO_2 in the slag usually increases, and CaO and FeO content decreases. Furthermore, the type of additions and hence variations in the quantity of slag affect the initial slag composition and its change in the process of blowing. In all cases of various additions investigated, slags contained considerable amounts of SiO_2 and MnO (up to 40-45% of either); the CaO content is small (up to 4%) in slags formed in the process of blowing pig iron without the addition of limestone, and it reaches 25-35% for pig iron with additions. As a rule, FeO content in the slag does not exceed 10%.

A comparison of different cases of additions shows that the least amount of tubing (300-400 mm per operation for phosphorus-containing pig iron and 600-700 mm for phosphorus-free pig iron) during the process of blowing is used up for mixed additions (ore and limestone, and ore and lime) and for the addition of ore only. In these cases, slag is formed prior to the commencement of blowing and hence the tubing gets well covered with slag. Various amounts and conditions of the slag, depending on the type of additions, affect the behavior of the slag during the process of blowing. In the cases where a large amount of ore was added, the slag sometimes foamed and overflowed the rim of the ladle; in other cases such phenomena were not observed.

In the case of surface blowing and addition of 1.5% each of ore and limestone, the utilization of the ladle constitutes 80-85%.

Blowing regime. The input and pressure of oxygen, the ratio of steam to oxygen in the mixture, and the depth of the oxygen tube immersion vitally affect the course of the oxidation of the inclusions in pig iron.

Oxygen input per ton of pig iron amounted to 4-6.5 standard cu m and depended on how high the ladle was filled, on the type of additions, and the course of the process. With the increase in the input of oxygen for

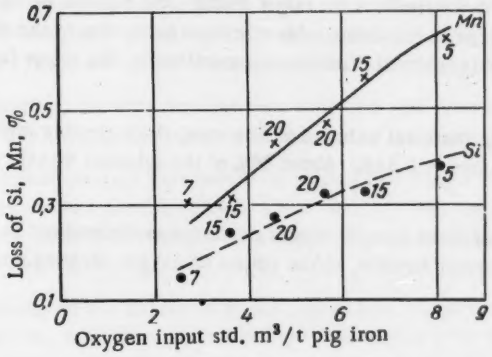

Fig. 2. The dependence of the removal of silicon and manganese (in 45 min of blowing) on the input of oxygen.

the blowing operation (Fig. 2) and with the increase in oxygen pressure, the removal of silicon and manganese increased. Steam consumption was 2-5 kg per ton of pig iron. The input of steam was adjusted in such a way as to prevent the evolution of brown fumes.

Mean oxygen : steam ratio (by weight) in the mixture of both, used for various cases of additions, was equal to 1.7-3.5 and depended on the composition of the addition, the moisture content of oxygen, the temperature of pig iron, etc. The greatest oxygen-to-steam ratio was observed in the operations involving large quantities of additions (4-5%). Apparently, this is due to an increased layer of slag and the cooling effect of the materials (limestone, ore) not assimilated by the slag before the blowing; both factors impede in formation and evolution of brown fumes and, hence, the ratio of oxygen to steam in the mixture of both can be increased.

In the test on surface blowing, when the end of the tube (3/4 inch diameter) was placed 150-200 mm above the slag level, the pressure of oxygen and steam before the mixer was about 3.0 atm higher than in the test of immersion blowing, and the input of steam-oxygen mixture per unit time per t of pig iron was approximately 25% lower. At the same time the rate of removal of manganese, silicon, and carbon was the same as in the case of the immersed tube. Oxygen: steam ratio was 3.2, compared with 2.5 for blowing with the immersed tube. This is explained by a smaller vaporization of iron and, hence, a more limited formation of brown fumes.

Tube consumption in the surface blowing constituted 100 mm for one operation as compared with 300-400 mm for blowing with the immersed tube.

Change in pig iron temperature. In the course of some operations, the temperaturature of pig iron was measured by means of platinum rhodium– platinum immersion thermocouple in the runner during the filling of the ladle, in the ladle (immediately after it was filled with pig iron), and prior to the blowing in the mixer building. The temperature of pig iron without any addition was, on the average, 41° C lower in the ladle than in the runner, and for pig iron with additions (1.5% limestone and 1.5% ore) it was 98° C lower in the ladle than in the runner. During transportation in the ladles, pig iron without additions was cooled 31° C, while pig iron with additions was hardly cooled at all (only 3° C)* in the ladle. The favorable insulating effect of the slag which has formed and of heat evolution from the continuing burning out of the impurities are noticeable. Hence, by the beginning of oxygen blowing, the pig iron in the ladles with additions was at a temperature only 29° C below the temperature of pig iron without additions.

The average increase in temperature during oxygen blowing varied between 25° C and 70° C for various tests. Tests in which the additions were used showed greater temperature rise during the blowing, which is due, apparently, to a greater burning out of silicon and to the protective effect of the slag in the ladle. At the same time, the cases in which additions were used are distinguished by a high oxygen-to-steam ratio.

Thus, in the tests in which additions (1.5% ore and 1.5% limestone) and surface blowing were employed, the temperature of pig iron prior to charging into the mixer was 40° C higher than the temperature of addition-free pig iron which was not blown through with oxygen.

In order to test the uniformity of pig iron composition, in 30 runs after oxygen blowing, samples were taken after about half of the metal was poured into the mixer. In addition, pig iron from three ladles was cast on the casting machine and 2-5 samples were taken during the process of casting. Analysis showed a uniform composition of pig iron in the ladles, this of course being due to the vigorous mixing of metal in the ladle during the blowing. Only sulfur is distributed nonuniformly, the upper layer usually having a higher content of sulfur.

Metal losses. From the material balance of four runs, the following metal losses were determined: splashes – 0.51%; iron loss in brown fumes – 0.04%. About 80% of the splashed metal can be returned to the ladle; hence, overall metal loss is 0.15%.

Hydrogen content. For some runs, hydrogen content was determined in the samples taken during the tapping of pig iron from the blast furnace, in the course of oxygen blowing, and at the charging of pig iron into the mixer.

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* Data on pig iron cooling during transportation were taken during the summer period.

On the basis of the analysis by means of the Morozov apparatus, it was established that hydrogen content increased from 2.3-3.9 cu cm/100 g to 4.9-6.0 cu cm/100 g during the process of blowing, and then decreased to 4.2-4.3 cu cm/100 g during charging into the mixer.

THE EFFECT OF THE CONDITIONS OF POURING ON THE QUALITY OF 5-6 t INGOTS AND OF STEEL PLATE ROLLED FROM THESE INGOTS

G. A. Chikalenko and E. V. Shevbunov

At the Il'ich Works, at present, the 5-6 t ingots of carbon steel and steel for boiler plate are bottom-poured into big-end-up molds which have a fixed bottom with two outlets (Fig. 1) and are provided with a hot top. The metal is poured from 130-t ladles through a 40-45 mm diameter nozzle. Four molds are placed on each bottom plate. The fountain is made of thick brick, the diameter of the inner runner being 50 mm. The metal is poured through two chamotte brick tapering nozzles of 60 mm outlet diameter (Fig. 2, a) which are inserted into the bottom of the ladle. The time for filling the molds is 4.5-6.0 min and for filling the hot top 1.5-2.0 min. Lunkerite which consists of 45% chamotte clay (grain size under 5 mm) and 55% coke fines (particle size under 8 mm) is thrown on the surface of the metal in the hot top.

With such a procedure, an early formation of crust on the surface of the rising metal took place and hence it was difficult to reduce significantly the lamination of the plates. Investigations have shown that this defect is caused by local accumulations of exogenous nonmetallic impurities. In order to prevent too early a formation of crust on the surface of the metal in the mold, it is necessary to increase the speed of the metal stream entering the mold; this can be achieved in two ways: either by increasing the rate of metal pouring (i.e. increasing the linear velocity of pouring) or by reducing the outlet cross section of the nozzles. If the first method alone is employed, there is a danger of the ingots being affected with cracks.

The inspection of a large number of ingots revealed that all the cracks found on the surface of the ingot are of the hot-crack type and can be classified according to the direction into longitudinal and transverse cracks. Transverse cracks are found mainly on the parts of the large sides of the ingot under the hot top and are of the type of cracks which are caused by the hanging of the ingot when the upper inner edge of the mold gets eroded or the hot top is not set up properly. The longitudinal cracks are found predominantly on the lower half of the large side over the middle third of its width. There are hardly any cracks in the upper half of the wall although some of the cracks extend over the whole height of the ingot. The distribution of the cracks on the large sides of the ingots, in the zone of the maximum effect of the steel stream on the ingot skin which is being formed, makes it possible to study the dynamic effect of the stream of metal entering the mold.

For this purpose, the tapering chamotte nozzles were replaced with nozzles which had a cylindrical opening 50 mm in diameter (Fig. 2, b), the outlet cross-sectional area of the opening being thus reduced by 30%.

The nozzles with cylindrical openings were tested on casting 5-6 t ingot of St. 3sp steel into molds provided with two feeders. Owing to the cylindrical form of the nozzle it was possible to obtain a more stable stream at a lower linear velocity and so to prevent, for a longer period, the formation of a hard skin on the surface of the metal.

Data relating to the quality of the surface of ingots poured into molds through cylindrical nozzles are given in Table 1. For comparison, the information relating to the ingots from the heats of the same period, but poured through tapering nozzles are also given.

For a better estimation of the surface quality of the two groups of ingots compared, data on both large sides of ingots (total number of ingot walls inspected was twice the number of ingots) are given in Table 1,

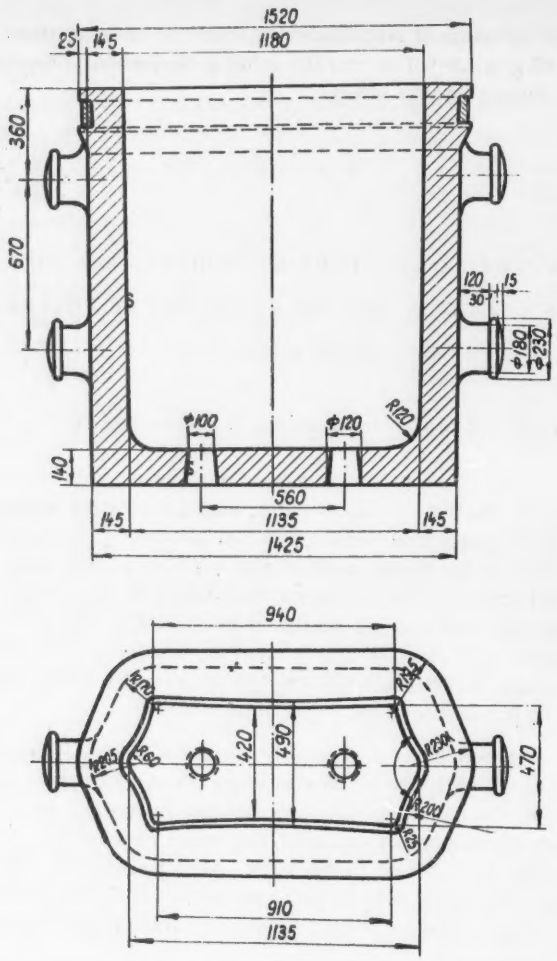

Fig. 1. Mold for 6 t ingots for steel plate.

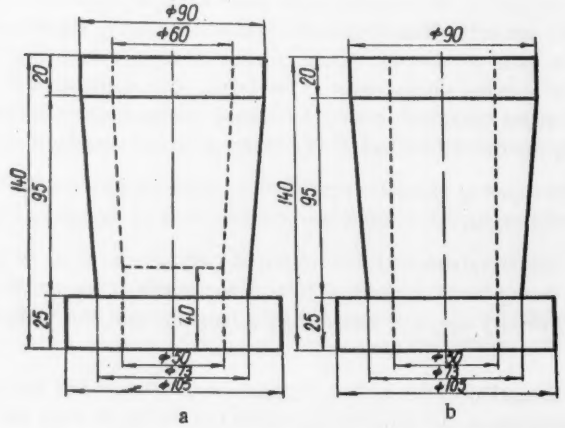

Fig. 2. Chamotte clay nozzle: a) with tapering opening; b) with cylindrical opening.

since it is possible that only one side may have surface defects.

It is seen from Table 1 that the change from the tapering to the cylindrical nozzle resulted in an improvement of ingot surface represented by a decrease in longitudinal and transverse cracks. Apparently, this is the result of a weakened effect of the stream of metal which forms crust on the ingot.

After the ingots were rolled into plates, the surface and the cut edges of the plates were thoroughly inspected. The ingots were rolled mainly into plates 25-40 mm thick. The quality of the plates rolled from ingots cast into molds with cylindrical and with tapering nozzles is represented by the data given in Table 2.

TABLE 1

Data on the Quality of Ingot Surface

Shape of the nozzle	Number of ingots cast	Number of large sides of ingots inspected	Among this number									
			Without defect		defective							
					longitudinal cracks		transverse cracks		refractory inclusions		folds	
			number	%	number	%	number	%	number	%	number	%
Cylindrical	425	850	640	75,3	104	12,2	72	8,48	30	3,55	4	0,47
Tapering	378	756	447	59,3	150	19,8	114	15,0	42	5,5	3	0,40

TABLE 2

Data on the Quality of the Plates

Nozzle shape	Number of ingots rolled into plates	Primarily rejected plates with defects of metallurgical origin		Plates rejected because of											
				longitudinal cracks		transverse cracks		seams		scabs		nonmetallic inclusions		folds	
		number	%	number	%	number	%	number	%	number	%	number	%	number	%
Cylindrical	434	122	28,1	17	3,9	38	8,8	43	9,9	14	3,2	8	1,84	2	0,46
Tapering	394	135	34,35	21	5,35	35	8,9	66	16,8	8	2,05	1	0,25	4	1,0

TABLE 3

Distribution of Seams Depending on Their Position on the Edges of the Plate

Position of seams on the edges	Number of plates with seams, % of total number inspected plates rolled from ingots cast through	
	cylindrical nozzle	tapering nozzle
End edge top	3,7	4,6
End edge bottom	3,7	5,3
Side	2,5	6,9
Number of plates inspected	434	394

Comparison of the data given in Table 2 shows that the reduction in longitudinal cracks on the large sides of the ingots resulted in an improvement in the quality of the plates with respect to this defect. However, the introduction of cylindrical nozzles had the most marked effect on the occurrence of seams on the plates, this defect being substantially reduced. The reduction in defects due to seams is explained by the fact that at the same linear velocity of pouring in the case of the application of cylindrical nozzles, the surface of the metal in the mold remained clear over long periods; i.e., the skin folding, which is one of the causes of seam formation, was prevented. In Table 3, a comparison of the distribution of seams on the cut edges of the plates is given.

Comparison of data given in Table 3 for both cylindrical and tapering nozzles shows that the replacement of the tapering nozzle by the cylindrical one resulted in a marked reduction in the amount of side seams.

Cylindrical nozzles have been adopted at No. 2 open-hearth furnace plant, for the bottom-pouring of ingots for plate rolling.

ADVANCES MADE IN THE PAST 10-12 YEARS IN THE REDUCTION OF THE INGOT HOT TOP

E. A. Iodko

The "Zaporozhstal" Works

The maximum shrinkage of liquid steel on solidification in the mold constitutes, according to literature, from 2 to 5.3% of the initial volume of metal. At the same time, the volume of metal poured into the hot top rarely constitutes less than 15% of ingot volume at the majority of Soviet and foreign works. Hence, about 10% of ingot weight solidifies in the hot top owing to poor insulating properties and inefficient design.

The amount of metal in the hot top can be reduced by an appropriate design of the hot top, an improvement of its heat-insulating properties, the heating of the metal in the hot top, and by establishing a higher pressure inside the hot top of the ingot for the compensation of the shrinkage of the ingot, not only with liquid metal but also with partly solidified metal, in the solid-liquid two-phase state.

If the hot top is appropriately designed its volume can be reduced without its function as the reservoir of metal being impaired.

The volume of the hot top is equal to the sum of the volume of the shrinkage and the volume of metal which solidifies in the hot top in the course of the complete solidification of the ingot. The amount of metal solidified in the hot top depends on the heat-insulating properties and the form of the hot top.

In recent years, several papers on the calculation of the volume of the hot top have been published. Only approximate data on the volume of the hot top can be obtained by means of calculation; this may sometimes be found adequate for work in the foundry where casting of very diverse forms are made, but it is not satisfactory for ingot casting in the steelmaking industry.

Some specific features of metal solidification must be taken into account for an appropriate design of the internal shape of the hot top cross section. It has been established that the metal in the corners of a rectangular hot top solidifies quicker than at the flat sides. Hence, it is better to make hot tops with cut-off corners or of a circular cross section. The volume of the hot top can then be reduced without the function of the hot top being impaired.

The layer of the metal solidified along the narrow sides of the rectangular hot top is bigger than the layer of the metal solidified in the same time at the larger sides. Hence, it is desirable to have, as far as

possible, the large sides wide apart from each other and the narrow sides as near to each other as possible.

For flat and square ingots, it is best to have multilateral or circular hot tops; but this shape results very frequently in a short life of the hot top and an inferior surface of the ingot.

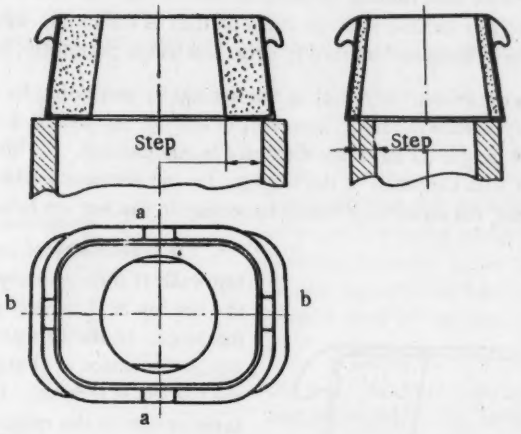

Fig. 1. Method of placing a circular hot top on the ingot for plate.

As is seen from Fig. 1, the step at the transition from the ingot body to the hot top, i.e., the difference between the internal dimensions of the mold and the hot top where they are in contact, may in fact become negative in the plane a – a, and positive in the plane b – b. Thus, the ingot may be hanging at the large sides and hence horizontal cracks may be formed, and at the narrow sides the hot top will be rapidly eroded. Therefore, one has to abandon the best shape of the hot top, gaining on the other hand on its increased life.

From the practice of several works, it has been established that the maximum step which can be allowed between the mold and the hot top, without a substantial decrease in the life of the hot top, is 45 mm. If a larger step is necessary, a so-called bipyramidal hot top (Fig. 2), whose internal shape consists of two truncated pyramids, is employed. The narrow sides of the lower pyramid are at a small angle to the horizontal, and the sides of the upper pyramid are nearly vertical. By means of such a hot top, it was possible to reduce the initial weight of metal in the hot top to 3% of the ingot weight without markedly lowering the life of the hot top.

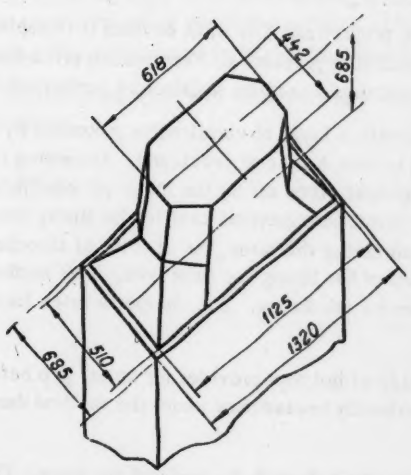

Fig. 2. Outline of the hot top of a 12.1 ingot at the Pittsburgh Works.

Successful tests of the bipyramidal hot top were carried out at the Makeevka Works. At the "Axovstal" Works, good results were obtained in tests of the hot top with variable oval cross section in the form of two truncated cones, the side of the lower cone being at a very small angle to the horizontal and the upper cone being at a very small angle to the vertical.

When killed steel is bottom poured, an abrupt transition from the ingot body to the hot top and a large step are undesirable, not only because the life of the hot top is reduced, but also because of possible folding of the ingot skin when it enters the hot top. In order to employ a circular hot top for a rectangular ingot, one has to provide for a gradual transition from the rectangular to the circular cross section inside the hot top itself. At the Stalino Metallurgical Works, a hot top has been adopted which in its lower part is of square cross section passing over into circular cross section at 1/3 of the height of the hot top. Patterns are used when the hot top is rammed.

At present, at several works, the hot tops are lined with chamotte brick, the procedure being considerably simpler than ramming. It is then advantageous to fit exchangeable cast iron plates on the narrow sides of the hot top for flat ingots, the worn plate being easily replaced when the hot top is relined (Fig. 3).

If the amount of metal cut off the top of the ingot is not determined by the depth of the pipe but by the extent of the defects resulting from skin folding (in bottom-poured ingots), then it is more appropriate to have a hot top with a larger inner cross section than the cross section of the mold; this is the practice at the Zlatoust Works. Since the ingot is then "hanging" uniformly over the whole perimeter, horizontal cracks do not appear.

A tendency is to reduce the amount of metal in the hot top by increasing its taper or by making it of spherical shape. A more pronounced tapering, however, of the hot top entails a reduced cross section at the top and, hence, an increased height (if the same capacity is maintained). This results in an increased surface of contact of the metal with the walls of the hot top, i.e., an increase in the amount of metal which is solidifying at the walls. Thus, the amount of metal necessary in the hot top is increased.

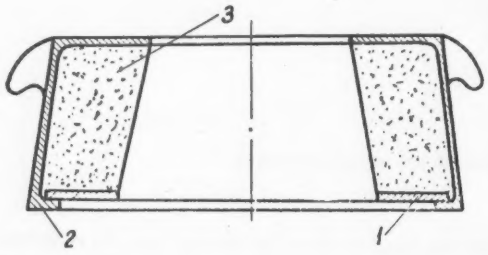

Fig. 3. Hot top with the replaceable cast iron plate: 1) replaceable plate; 2) casting; 3) lining

It is economical to increase the taper of the hot top walls if there is a superflous amount of metal in the hot top or if the top end of the ingot is not adequately insulated. In the first case, very often it does not matter which dimension is changed, provided that the volume of the hot top is reduced. In the latter case, the increased taper results in the reduction of free surface of the metal, where heat is lost at a high rate, and in the reduction of the amount of useless metal solidified in the hot top.

Apparently, it is advantageous to allow a minimum taper of the hot top (8% to 9% on each side) which would make possible the lifting of the hot top without damaging the lining.

The ratio of the height of the hot top to its representative horizontal dimension is determined, first of all, by the efficiency of the insulation of the top end of the ingot and the side walls of the hot top. Sometimes, heat losses through the top end of the ingot are negligible compared with the losses through the side walls. This is usually the case when the free surface of the ingot is strewn with lunkerite and covered with insulating lids. It is then economical to have a possible lower level of the metal in the hot top, since in this way heat losses from the metal are reduced and the volume of the hot top is more efficiently utilized.

By improving the heat-insulating properties of the hot top and designing it appropriately, it is possible to reduce its volume to 8-10% of the volume of the ingot without changing its cross section.

There are several ways of improving the heat-insulating properties. The most obvious is to replace the chamotte brick lining with another material of better heat-insulating properties. Vermiculite brick and "Lapex" brick can be used. Lining made of those bricks lasts for 17 teemings – a quite satisfactory performance.

In England, recently, a new design of the hot top, lined with a layer of vermiculite protected by a 3-mm-thick steel plate onto which a 6-mm-thick refractory coating is laid, has been developed. According to data published in literature, the refractory lining absorbs 50% of the heat given off by the ingot on solidification. Therefore, in order to reduce heat losses it is desirable to use brick with internal cavities for lining the hot tops, as in this way the weight of the hot top is reduced, its size remaining the same, the amount of absorbed heat is also reduced and, at the same time, the insulating properties of the lining are improved. This method, however, has not become popular as it requires special refractories for the lining. The chamotte brick for mold fountain lining can be used for lining the hot top (Fig. 4).

At the KMK (Kuznetsk Metallurgical Combine) the design of hot tops provides for an air gap between the lining and the metal casing, but this manner of lining can hardly be classified under the method described above.

About 20% of the heat of the metal in the hot top is dissipated through the body of the ingot. Therefore, a chamotte diaphragm with a small opening can be inserted between the mold and the hot top. By this

means in some cases it was possible to substantially reduce the weight of metal in the hot top. This method, however, has not been generally accepted, as the placing of the diaphragm constitutes an additional operation.

Apparently, an efficient method of reducing the dead head of ingot is to heat the metal in the hot top; in this way, the weight of metal in the hot top can be reduced to 5% of ingot weight. Heating with a gas burner is not satisfactory. Electric arc and induction heatings can be employed only for high-quality steel, for which the cost of electric power is economically justified and recoverable. Therefore, more economical sources of heating are being explored.

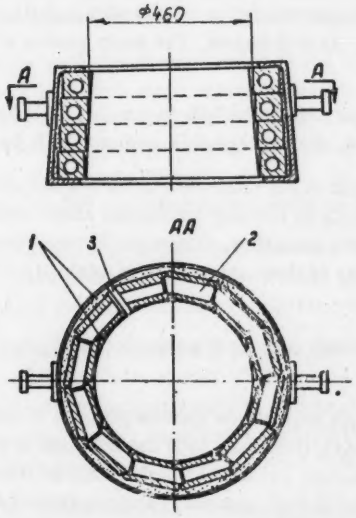

Fig. 4. Hot top lined with fire brick for mold fountain: 1) refractory material; 2) fountain brick; 3) wedge.

Trials were made to burn powder of 75% ferrosilicon in an oxygen stream by the method of Czech metallurgists. The test gave satisfactory results but it is impossible to employ this method for a large quantity of ingots owing to the bulky equipment required. The attempt at heating the metal in the hot top by blowing oxygen through it, was unsuccessful because the manganese content in the middle zone of the ingot top was markedly reduced.

It seems that the most promising method of heating is by means of exothermic materials, containing fuel and an oxidizer, e.g., saltpeter or cinders. These materials can be thrown on the surface of the metal in the hot top and can be used for lining the hot top; an insert of exothermic material can also be placed in the hot top.

High pressure in the hot top is hardly made use of in the steelmaking industry but it is frequently employed in foundries. A ceramic cartridge containing a charge, usually of chalk, is inserted into the feedhead. The deadhead is desirably of a spherical shape. The metal in the deadhead fairly quickly forms a thick skin which does not allow the escape of the gases formed as a result of the gas-producing charge. In the steelmaking industry, this method is rarely employed owing to the difficulties in inserting the cartridge and the danger of hot metal splashes. According to reports in literature, the weight of metal in the hot top can be reduced by this method to 10% of ingot weight.

The above considerations show that the best method of reducing the deadhead of the ingot is an appropriate shaping of the hot top. This method involves least expenses and results in substantial savings. The next steps are the improvement of thermal insulation of the hot top by means of hollow brick and the use of exothermic coatings. Electric arc and induction heating can be successfully employed at electric steelmaking plants.

At the same time, the search for new ways and means of reducing the weight of the ingot deadhead must be continued.

THE PRODUCTION OF OPEN-HEARTH PHOSPHATE SLAGS AT THE "AZOVSTAL" WORKS

F. F. Sviridenko and A. N. Popova

In 1955, a slag-grinding section of the open-hearth furnace plant at the "Azovstal" Works of 340,000 t annual output of ground slag was put into operation and the production of phosphate slags for fertilizers was commenced.

A low silica content in the charge materials promotes an efficient oxidation of phosphorus during the melting period and the production of slag rich in phosphorus anhydride. As it is known, the main sources of silicon are liquid pig iron and iron ore.

If the silica content in the ore is increased by 1%, the average content of P_2O_5 in the slag falls by 0.5-0.8%; and if the silicon content in the pig iron is increased by 0.1%, the average P_2O_5 content falls by 1.5%.

It has been established that when the silicon content in pig iron is not more than 0.5% and the silica content in the ore not more than 8%, the concentration of soluble P_2O_5 in the slag constitutes about 14%. The increase of acidic oxides (SiO_2; P_2O_5; Al_2O_3 and Fe_2O_3) increases the solubility. These oxides are interchangeable with regard to their effect on the solubility; 1% increase in any of them increases the solubility of P_2O_5 by approximately 5.5%.

The increase in the content of basic oxides (CaO, MgO and MnO) reduces the solubility of P_2O_5; 1% increase in any of them causes a 2.7-3.3% fall in the solubility of P_2O_5.

The effect of the ferrous oxide content on the solubility of P_2O_5 depends on the composition of the slag. At low values of the total concentration of SiO_2, P_2O_5, Al_2O_3 and Fe_2O_3 (less than 34%) the increase in FeO content increases the solubility. In this range, ferrous oxide acts like acidic oxides. At high values of the total concentration of SiO_2, P_2O_5, Al_2O_3 and Fe_2O_3 (over 34%), the increase in FeO content is accompanied by a decrease in the solubility of P_2O_5. In this case, FeO behaves like basic oxides. Hence, settling the slag in the slag ladles with basic materials reduces the solubility of P_2O_5 in the slag, so it is better to use acidic materials or water. Since the solubility of P_2O_5 depends on the particle size of the slag, the sample for the determination of solubility is always taken from the slag which has passed through the sieve of 0.175 mm apertures. The solubility is determined by the method developed at the Ukrainian Institute of Metals and the "Azovstal" Works

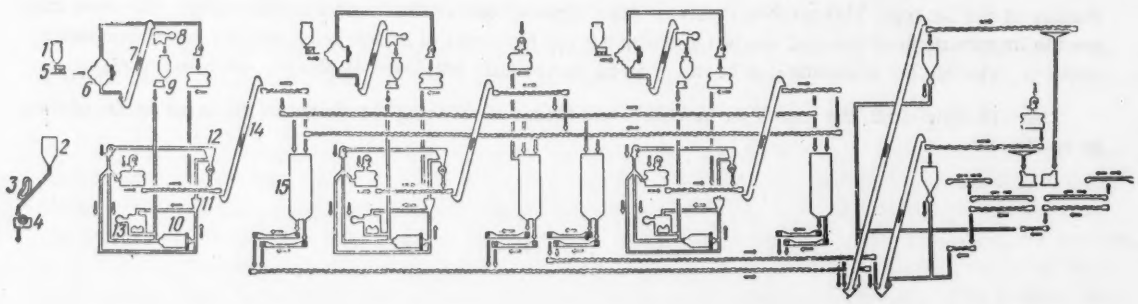

Diagram of slag-grinding section of the "Azovstal" Works: 1) and 2) feed bins; 3) chain feeder; 4) jaw crusher; 5) car feeder; 6) coarse mill; 7) bucket elevators; 8) separators; 9) table feeders; 10) fine mills; 11) air separators; 12) cyclons; 13) electro-magnetic separator; 14) bucket elevator; 15) silos.

The slag ladles, with the slag, are conveyed to the slag-grinding section and are discharged on the slag yard. Unloaded slag is crushed with drop weights operated by the overhead magnetic grab cranes. Crushed

slag is then transferred by grab cranes to covered storage floor which is provided with one grab crane and a second magnetic grab crane. The slag is then charged into coarse mill (see Figure) as required. The feed bins are covered from above with a wire screen of 180 × 180 mm mesh to prevent falling of too-large lumps into the mills.

Slag lumps which remain on the screen are fed via a bin and a chain feeder into the jaw crusher.

Slag from the bins is transferred by car feeders to the coarse mills which have product outlet at the circumference. Coarsely ground slag is transferred by bucket elevators through magnetic separators to the feed bins of fine mills. The slag is charged from the bins to the mills by means of table feeders. From mills, the slag is removed into air separator by means of air from a high-pressure air fan. Inadequately ground slag settles in the separators and the smaller size fraction is taken by the air into the cyclons and settles at the bottom.

Large particles of slag settled in the air separator are returned to the fine mills through electromagnetic separators. Dust-free air is drawn by the fan and is returned back into the mills, which are operated under vacuum to prevent dust dispersion. Air surplus is blown out to the atmosphere through a system of leaf filters. Ready powder from the cyclons together with the powder accumulated in the filters are transported by a screw conveyor to bucket elevators. From the elevator, the powder is distributed by screw conveyors to four silos of 8500 cu m capacity. 15,000 t of phosphate powder of 1.8 t/cu m bulk weight can be stored in the silos.

The phosphate powder from the silos enters two collecting screw conveyers arranged along the silos, passes through chain dischargers into two bucket elevators, hence into a sieve screw, and then through a collecting feeder to a packing machine equipped with three taps. There are two packing machines at the plant.

The phosphate powder is packed into paper bags which have, in one corner, a small inlet with a valve which is automatically closed after the bag is filled. Filled bags fall onto two belt conveyors passing through the length of the packing section. The bags are then transferred onto two transverse conveyors which load them into the cars. Powder from torn bags falls into screw conveyors, placed in a duct under the belt conveyors, which take it back through the transfer screws, bucket elevators and sieve screw to the packing machines.

The whole complex – mills, screws, silos and packing machines – is under vacuum and connected to dust-removing filters.

Material balance was carried out for several heats in order to determine the yield of the slag. Ore containing 7% SiO_2, and pig iron containing 0.4% Si were used for steelmaking.

Data on the distribution and the composition of the slag in different periods of the heat are given in the table.

Mean Amount of Slag and Its P_2O_5 Content

	Slag yield		P_2O_5 content, %		% dissolution
	tons	% by weight of steel	total	soluble	
In course of melt. period	33,4	9,4	14,3	12,8	90,0
Before melting and at beginning of the refining period	26,7	7,5	14,4	11,5	80,0
Total conditioned* slag	60,1	16,9	14,35	12,7	88,7
Conditioned slag in course of refin. period	26,4	7,4	—	—	—
Total	86,5	24,3	—	—	—

* The slag which contains 10% or more of P_2O_5 is regarded as conditioned slag.

It is seen from the data in the table that the amount of conditioned phosphate slags constitutes 16.9% of the weight of steel produced. If the scrap and the reguli which are removed in the course of dressing and grinding are subtracted from this figure, the amount of conditioned phosphorus-containing slags in the experimental heats is 14.2%. In actual fact, a considerably smaller amount was obtained:

	Years	
	1956	1957
Phosphate slag (powder), % by weight of steel produced	8.8	9.4
Average content of soluble P_2O_5, %	12.97	11.85

The causes of this discrepancy are:

1) a higher SiO_2 content in the ore (9.2% on the average, in 1957) and Si content in the pig iron (0.6% on the average, in 1957); this resulted in the reduction of P_2O_5 content in the slag and a lower yield of conditioned slags;

2) the loss of some of the conditioned slag due to the overloading of the furnaces by exceeding the design capacity;

3) a low phosphorus content in the pig iron (below 1.5%), as it was necessary to use some quantity of phosphorus-free pig iron during the periods of a high rate of production;

4) an inadequate separation of conditioned and nonconditioned slags during the filling and transportation of slag ladles.

THIRD MEETING OF THE STEELMAKING INDUSTRY SECTION OF THE PERMANENT COMMISSION FOR FERROUS METALLURGY OF THE COUNCIL FOR MUTUAL ECONOMIC ASSISTANCE

In April, 1958, the third meeting of the Steelmaking Industry Section, jointly with the Section of Refractories, of the Permanent Commission of the Ferrous Industry of the member countries of the Council for Mutual Economic Assistance took place in Dnepropetrovsk. Specialists on steelmaking and refractories of the People's Republic of Bulgaria, the People's Republic of Hungary, the German Democratic Republic, the People's Republic of Poland, the People's Republic of Rumania, the USSR, and the Czechoslovak Republic took part in the conference. The conference heard and discussed the reports of specialists from the countries of the Soviet block on carrying out the resolutions passed at the Paris second meeting of the Section and relating to the increase of steel output in 1958-1960, the working out of basic policy for the development of new techniques in the steelmaking industry in 1960-1975, the improvement of the quality of refractories used in the steelmaking industry, the development of the proposed plans for increased production of roll-bearing steel, and on the standardization of industrial specifications.

At the plenary session, the reports of the delegations from the countries represented on the work carried out in the field of steelmaking in the period from October, 1957, to April, 1958, were presented.

The delegations reported the achievements of metallurgists in the socialist countries in 1956-1957, and discussed the tasks of the steelmaking industry for the next few years with respect to raising the quality and the quantity of steel production. The steelmaking Section noted that as a result of carrying out the recommendations passed in 1957, the cooperation between the countries improved, an even closer contact was established between the steelmaking industry workers, and the exchange of experience and information became more extensive.

It should be mentioned that the reports of all the delegations on the work carried out in 1956-1957 were valuable contributions to the exchange of experience between the countries.

It was recommended that a further increase in steel output in 1960-1975 of the member-countries of the Council for Mutual Economic Assistance should be aimed at by means of increasing the capacity of the main equipment, by the extensive use of oxygen in all metallurgical processes, by the use of cold high-calorific fuel which does not require preliminary heating in the regenerator, by an economical utilization of available natural gas in the steelmaking industry, by building new open-hearth and electric arc furnaces and large capacity Bessemer converters as well as by increasing the capacity of existing steelmaking units, modernizing them, increasing steel-pouring equipment, adopting continuous casting, and changing some of the furnaces to double charging.

The principle method of steelmaking in the socialist countries will remain the open-hearth process in large-capacity furnaces. At the same time, taking into account the fact that steel obtained by treating conversion pig iron with oxygen should approach the quality of open-hearth steel (in particular low-carbon rimmed steel), the industry should expand converter-steel production with the application of oxygen.

With a view to extending the service life of steelmaking units, it is recommended to change over to new types of high-quality refractories. The Steelmaking Section has determined basic specifications for refractory materials for open-hearth furnaces (Table 1).

TABLE 1

Properties of Refractories for Open-Hearth Furnaces

Property	Magnesite-chromite (heat-resistant)	Periclase-spinellide
Beginning of the deformation under the load of 2 kg/sq cm, °C	1550	1600
Apparent porosity, % not more than	17	14
Heat-resistance (heating to 850°C), heat change, not less than	100	100

TABLE 2

Chemical Composition (sample from the ladle) Proposed Steel Grades, %

Steel grade	C	Mn	Si	Cr	P	S	Ni	Cu	Ni+Cu	Recommended application
						not more than				
№ 1	0,95—1,10	0,30—0,50	0,15—0,35	1,30—1,65	0,027	0,020	0,30	0,30	0,50	Rings of not more than 14 mm thickness, bearing rolls and balls not more than 22 mm thick
№ 2	0,95—1,10	0,90—1,20	0,40—0,65	1,30—1,65	0,027	0,020	0,30	0,30	0,50	Rings of not more than 14 mm thickness, bearing rolls and balls not more than 22 mm thick

Remarks: 1. The permissible content of sulfur in steels made in induction and open-hearth furnace is not more than 0.03%.
2. Ball-bearing steel is produced in the USSR in conformity with GOST.*

For the upper part of the checker of the air and gas regenerators, the use of high-alumina brick (more than 42% Al_2O_3) of not more than 18% porosity was recommended.

*GOST All-Union State Standard.

The manufacture of high-quality alloy steels and alloys in induction and direct arc furnaces by the application of a high vacuum was recommended. A radical improvement in preparation of scrap for steelmaking furnaces should be achieved by supplying the steel industry with large-capacity fagoting presses and equipment for sorting and preparing the scrap. At the same time, a maximum possible mechanization and automation of production processes should be carried out, the spectrum analysis of the chemical composition of the metal and slag in the course of the process should be adopted, vaporization cooling of furnaces should be introduced, and waste-heat boilers should be installed.

Interesting recommendations were given with regard to the standard specifications for ball-bearing steel production.

The socialist countries will in the period up to 1965 change gradually to the production of all ball-bearing steel in electric furnaces, employing steel evacuation either in the ladle or on pouring. The Section recommended that the marking of steel grades as accepted in individual countries should be retained. Steel for export to other member-countries of the Council for Mutual Economical Assistance should be classified into two grades, Nos. 1 and 2, depending on its chemical composition (Table 2).

The amount of ball-bearing steel conforming to higher specifications with respect to microstructure should be determined by the Steel-Rolling Section, since the properties of this steel depend mainly on the vacuum treatment and thermal treatment of the metal.

In order to satisfy the specifications of the machine-building industry with respect to nonmetallic inclusions, it was considered desirable to introduce a special production of steel whose mark (number) with respect to nonmetallic inclusions should be within limits specified by GOST and by the consumers concerned. This steel should conform to the special conditions in accordance with the agreement made between the producer and the consumer.

On May 15, 1958, official standard specifications for ball-bearing and roller-bearing steel (GOST 801-58), to be officially introduced April 1, 1959, were approved. These specifications cover chromium steel and chrome-silicon hot-rolled and cold-drawn steel for ball and roller bearings. With respect to chemical composition, the specifications also cover wire, plate, strip, and pipe production as well as forging and die stamping.

TABLE 3

Chemical Composition of Ball- and Roller-Bearing Steel according to GOST 801-58

Steel grade	Chemical composition, %								
	C	Mn	Si	Cr	S	P	Ni	Cu	Ni + Cu
					not more than				
ShKh6	1,05—1,15	0,20—0,40	0,17—0,37	0,40—0,70	0,02	0,027	0,30	0,25	0,50
ShKh9	1,00—1,10	0,20—0,40	0,17—0.37	0,90—1,20	0,02	0,027	0,30	0,25	0,50
ShKh15	0,95—1,05	0,20—0,40	0,17—0,37	1,30—1,65	0,02	0,027	0,30	0,25	0,50
ShKh15SG	0,95—1,05	0,90—1,20	0,40—0,65	1,30—1,65	0,02	0,027	0,30	0,25	0,50

The ball bearings and roller bearings are made of ShKh15 and ShKh15Sg steels produced in electric furnaces and acid open-hearth furnaces. The chemical composition of ball-bearing steel as specified by the new GOST is given in Table 3.

The steel industry of the socialist countries will take the necessary measures, in the period up to 1960, for an improvement in the quality of ball-bearing steel with a view to changing to the production of this steel in conformity with the improved, more rigid specifications in the course of the subsequent two years.

The extremely friendly atmosphere of the conference should be mentioned. The concentrated creative efforts of the representatives of different nations forged a strong friendly bond between us, and this fact remains one of the main results of the Dnepropetrovsk conference on steelmaking. As confirmed by the unanimous

opinion of the members, the efficient organization of the conference contributed to its successful work. The members of the conference were provided with the best conditions for work and opportunities for studying the metallurgical works of the Dnepropetrovsk Economical Region.

The members of the Steelmaking Section in the Permanent Commission for Ferrous Metallurgy of the Council for Mutual Economic Assistance express their thanks to the chairman of the Dnepropetrovsk National Economic Council, N. A. Tikhonov, and the head of the Technical Production Department of the Council, D. G. Ignatenko, for their effort to make the conference a success.

As the conference was held at a big center of the metallurgical industry, the delegates to the conference had the opportunity of studying operational experience of local works and the adoption of technical improvements in production processes. The members visited the Petrovskii, the Krivoi Rog, and the "Zaporozhstal" Metallurgical Works. The director of each works personally aquainted all members of the conference with the steelmaking production processes at each works.

The many-sided collaboration among the socialist countries, based on the principle of complete equality, mutual respect of national interest, and mutual socialist assistance, successfully promotes the cause of socialism and communism and helps in making full use of the advantages of the world socialist economic system for the development of productive potentialities of all socialist countries and for the strengthening of the economic power of the whole socialist block.

M. A. Pertsev

Leader of the Soviet Delegation of Steelmaking Section of the Permanent Commission for Ferrous Metallurgy of the Council for Mutual Economic Assistance

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AN EFFECTIVE METHOD FOR ROLL RAGGING

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K. S. Gudovshshikov and Iv. A. Shapiro

The Zhdanov Metallurgical Institute and the "Azovstal" Works

The hatching, welding on, and knurling of rolls greatly improve their gripping power. These methods of roughening the surface of new rolls are especially important for the period up to the appearance of an erosion network, i.e., for the first 3-4 days of rolling. But these methods are not all equally effective.

The 1150 blooming mill at our Works is provided with forged rolls made of 55Kh steel; since 1949, the surface of the roll barrel and the first pass were hatched by means of a pneumatic chisel. The hatching had the form of short indentation lines 20-30 mm long, up to 3 mm deep, and parallel to the axis of the roll; in addition to the hatching a zigzag groove 12-15 mm wide and 3 mm deep (Fig. 1) was cut in the first pass. A worker could rag one roll with the pneumatic chisel in 12 hours.

However, the effect of the hatching was not satisfactory. Some improvement in roll bite was observed over the first two or three shifts of operation and then the edges of the notches began to draw in, the surface became smooth, and slippages took place again. To prevent this happening, it was even necessary to reduce drafts and add two passes. Only after the appearance of a coarse erosion network did the grip of the roll improve again.

In view of the disadvantages of this method of roll ragging, it was found economical to preserve the erosion network on the roll barrel and on the bottom of the passes (grooves) by automatic welding-on of the worn sides of the passes. The first time, the periodically welded-on rolls worked for 45 days (3 changes) without being machined down during intermediate machining. It was found, however, that the circular cracks extended in some places (most of all on the roll barrel) to the depth of 25-30 mm, impairing the strength of the rolls. The circular cracks, following the erosion network, develop especially rapidly on the third roll change (30-45 days of operation). Therefore, the erosion network should be kept for not more than one month (2 changes) and after that time the rolls should be machined down (15-20 mm).

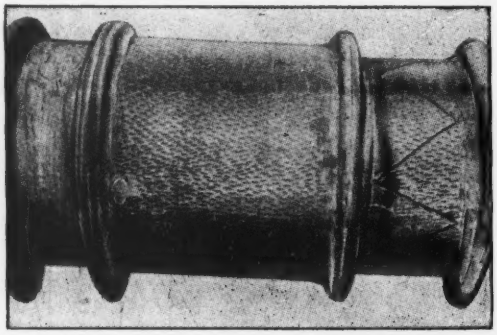

Fig. 1. Roll with hatched surface and cut zigzag grooves

A more frequent machining of rolls and the reduction of roll diameter considerably reduce the economic advantage of welding on the collars. In addition, during the first 3-4 days after the machining, the rolls do not grip the rolled piece well and hence a drop in the output of the blooming mill results.

Attempts were made to automatically weld longitudinal bands onto the roll barrel and, in the first pass, on the freshly machined surface The bands were 15-20 mm wide, 3 mm high and at a distance of 75-80 mm from each other. The gripping of the rolled piece was very good but the mill was subjected to vibration which had a damaging effect on the spindles, pinions, and main motors. Deep cracks appeared at places of welding-on. Furthermore, the indentation marks on the surface of the blooms, in particular of rail steel, were injurious to the surface of the finished product.

In recent years several methods, employed at other steel works, of ragging the surface of steel rolls of roughing mills were studied at the "Azovstal" Works. At the Kirov Works, a ragging roller with teeth arranged in rows are used; at the KMK, the teeth on the roller are arranged in staggered order. Similar ragging rollers (Fig. 2) for producing up to 3-mm-deep indentations on the surface of the rolls were made. It was found from observations that the indentations also begin to smooth out after 2 or 3 shifts of roll operation. In addition, as a result of cold hardening during ragging, the formation of the erosion network is delayed and, hence, the intermediate period of poor gripping is even longer than in the case of roll chiseling.

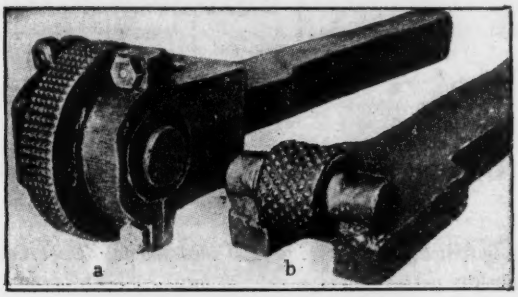

Fig. 2. Rollers for producing indentations on the surface of rolls: a) at the Makeevka Works; b) at the Kuznetsk Metallurgical Combine.

Very good results on the improvement of roll grip and a long preservation of the roughness of the surface were obtained when a special method of roll ragging, by producing projections on the roll, was applied. By means of a toothed cutter (Fig. 3) circular bands were cut on the bottom of the pass; the ridges were deformed with a 50-80-mm-wide toothed ragging roller made of 6KhVS steel. Owing to the application of special 100-mm-wide toothed cutters of a concave shape corresponding to the convexity of the bottom of the box passes, the rolls could be machined very quickly. A roll-lathe operator can machine all the passes and the barrel of a roll of the blooming mill in 3 hrs. Six to eight rolls can be ragged with one ragging roller. The working surface of the barrel and the bottom of the passes are then uniformly covered with pyramid-shaped projections, 2.5 mm high and of 5 mm x 5 mm base.

The study of the change in the surface of ragged rolls, by taking lead impressions every 24 hrs of roll operation, showed that the pyramidal projections wear very slowly and only get blunt gradually. It was found that, even after 12 days of operation, the lower part, about 1 mm high, of the projections on the barrel and in the first pass was still preserved. The formation of the erosion network is very much delayed and the network is oriented along the edges of the projection bases. It can actually be assumed that the necessary roughness of roll surface is maintained during the whole time that the rolls are in operation and, hence, increased drafts can be applied without danger of slippage.

At present, the rolls of the blooming mill and other roughing stands at our Works use ragging in the form

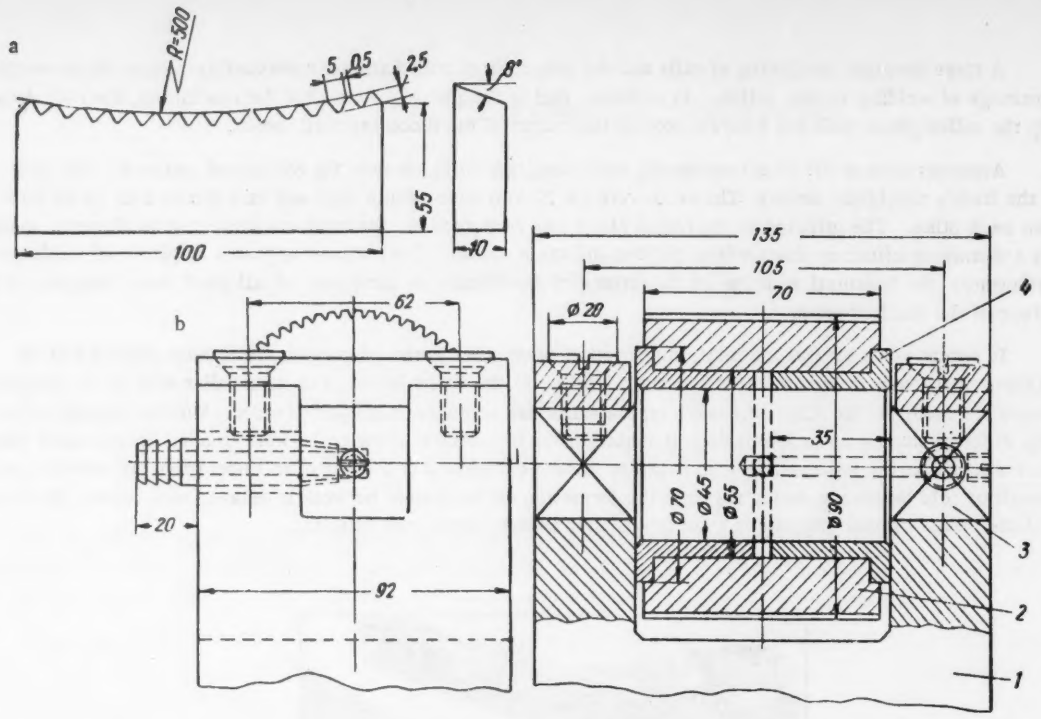

Fig. 3. Equipment for roll ragging: a) toothed cutter; b) ragging roller; 1 – holder; 2 – ragging tool; 3 – axis; 4 – bush

of pyramid – like projections. The output of the blooming mill remains on the same high level throughout the whole operation, and the drafts in the passes increase to 120 mm.

MECHANIZATION OF SAMPLE DELIVERY FROM THE HOT SAWS TO THE 800 MILLS

V. V. Tatarnikov, M. I. Rozhin and P. A. Chekhovskii

Rail-structural plant of the "Azovstal" Works

The samples from the hot saws were previously delivered manually to the 800 mill (120 m distance). It was an arduous and dangerous operation which somewhat hampered the output of the mill.

The delivery of the samples has now been mechanized; a special conveyor has been set up between the mill and the hot saws. On a long rail (see figure) fixed with cantilever brackets to the columns of the building, a trolley is suspended and a cable is firmly tightened to he trolley cleat by means of a screw yoke. When the rope is pulled, the trolley, together with the sample cage rigidly attached to it, moves on the rail. The rope is pulled by an electric motor via a reducing gear and a system of pulleys. The pulleys and the rope are positioned in such a way that the trolley can be moved in the direction towards the control desk of the foreman and then towards the hot saws.

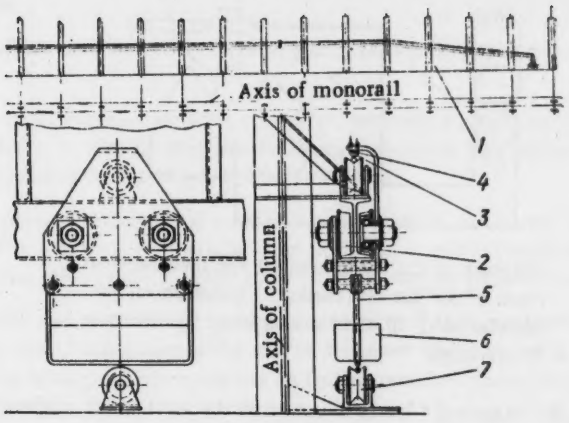

Diagram of sample trolley: 1) monorail; 2) trolley; 3) screw yoke; 4) rope; 5) suspension arm; 6) sample cage; 7) roll

Guide rolls for the rope are positioned over the whole length of the monorail. The motor is mounted on a special bed plate near the hot saw section.

EQUIPMENT FOR STRENGTHENING THE TEETH OF HOT SAWS

V. V. Tatarnikov, A. N. Zannes and F. P. Rybaev

The "Azovstal" Works.

In 1956, two types of equipment for the automatic hardening of the teeth of rotary hot saws for steel cutting were tested in operation at the "Azovstal" Works.

One apparatus was constructed on the basis of the machine for grinding the teeth of circular saws. The machine was adopted for electric heating of the teeth for hardening.

The second type of grinding and hardening equipment consisted of two units. One was designed for milling the saw teeth and the other for the electric heating and hardening of the hot saw teeth. Both units were driven by one electric motor.

The basic electric circuit is given in the figure. The characteristics of the step-down transformer are: primary voltage =380 v; secondary voltage =6 v; current of the secondary coil =1100 amp; power =6.6 kva.

The termini of the transformer low-voltage coil are attached to the saw, which is to be subjected to sharpening, heating, and hardening, and to the electrode holder, which contains a graphite electrode.

After the electrode is lowered on to the tooth the current is automatically switched on and the tooth is heated up.

In 4 seconds (the exposure time is controlled by time relay) the tooth is heated to the hardening temperature of a 3-mm-deep layer at the end of the tooth which is of 45-52° angle and 7-10 mm thick. The heat from the heated up zone is dissipated into the body of the saw and to the surroundings.

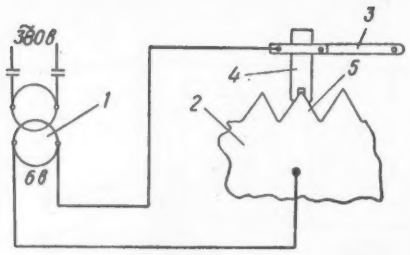

Diagram of the equipment for the thermal treatment of the hot saw teeth: 1) transformer; 2) circular saw; 3) electrode holder; 4) electrode; 5) saw tooth

The tests carried out at the "Azovstal" Works showed that the service life of the circular saws whose teeth were subjected to the strengthening treatment by means of electric heating and subsequent hardening, as described above, was 3 times longer than the service life of non-hardened saws.

MECHANICAL TILTING OF RAILS

V. V. Trofimov

OTK Engineer at the "Azovstal" Works

For a long time the railroad rails at our Works were tilted manually; the operation required a large number of workers and could not ensure a regular operation of the plant. When normal-length (12.5 m) light rail of R-43 type were produced at the Works, the disadvantages of manual turning were not very apparent; but with the change-over to the production of 25-m-long rails and the adoption of R-50, R-65 and R-75 heavy-type rails the mechanization of tilting operations became imperative.

At present, the rails are turned mechanically at the following five points:

1) prior to being charged into the retarded-cooling furnaces, the rails are turned from the horizontal position: "on edge" to the vertical position: "head up;"

2) prior to entering the cambering machine, the rails are turned "head up;"

3) after leaving the cambering machine the rails are turned "on edge;"

4) in the process of straightening on gag presses, the rails are turned into the position required for the elimination of a local or an overall curvature;

5) in the hardening chambers the rails are turned "head up" before hardening and "on edge" after hardening.

The apparently superfluous turning of the rails "on edge" after they leave the cambering machine and after hardening in the hardening chambers, was necessary in order to prevent mechanical damage to the base of the rail. When the rails are moved laterally on the gantries by means of dragging equipment, transverse scratches appear on the rail base. These scratches on cold rails constitute zones of stress concentration and lower the mechanical strength of the rails in service. Polishing the lubrication of sliding surfaces on the gantries did not ensure the prevention of scratches; and, therefore, before being dragged the rails were turned "on edge".

Incidental minor scratches on the flange of the base or on the side of the rail head had no effect on the mechanical strength of the rails.

The description of a few mechanical methods of tilting rails (which methods could be adopted at other works) is given below.

Turning hot rails. F. V. Popenko, an OTK controller, proposed a simple and original design of springboard-type turning fingers (Fig.1). On the diagram the finger is shown in its working position. 10-15 fingers can be set in one line (depending on the length of the rails rolled).

The hot rails are delivered in horizontal position by the transfers, are moved up the inclined plane of the finger, and turn through 90° under their own weight when falling in a vertical position or to the gentry. The rails are then assembled into piles by the transfers to be charged into the retarded-cooling soaking pits.

The lower edge of the end of the finger beam has to be cut off at an angle in such a way as to prevent the base flange of the rail from hitting against the end of the beam when the rail is turning through 90°. Under these conditions, the freely falling rail comes to rest on its base which is protected from dents that are easily produced in hot metal on impact.

Springboard-type turning fingers were found very satisfactory in operation. They can be effectively applied for turning rails of any type and length. The construction of the finger is simple and reliable.

The mechanization of the rail-turning operation resulted in a saving of 100,000 rubles annually, in respect to wages alone, owing to the reduction of operating personnel.

Moreover, as a result of the change-over from manual to mechanical turning of rails, it was possible to turn rails at a higher temperature (700-750° C). It was impossible to turn the rails manually at such a temperature, and the operators started to turn the rails when they cooled down to about 550° C.

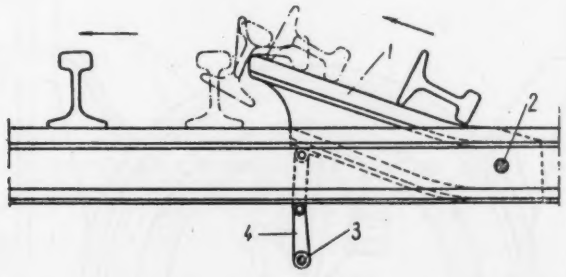

Fig. 1. Springboard-type tilting finger: 1) finger; 2) axis of rotation of the finger; 3) axis of rotation of the lifting mechanism; 4) hinged link

The following advantages were gained by the fact that the rails could be turned and charged into the soaking pit at a higher temperature:

1) the charging of rails into the pits was made easier since hot rails have a smaller tendency towards longitudinal bending;

2) the cases of rail twisting which occurred during manual charging ceased completely (at about 600° rails are sufficiently plastic to get twisted under the force applied by the workers turning them);

3) the mechanical turning assists in reducing defective rails, which were due to the thermal regime not being strictly maintained at the commencement of charging ; when the rails are turned mechanically the actual temperature of the rails during charging into furnace is usually considerably higher than given in the specifications. If the rails are cooled, by whatever causes, below the specified temperature, a large proportion of the rails produced have defects resulting from the retarded-cooling regime not being maintained. The heat stored in rails turned at a temperature of 700° C is found to be adequate for the case of any possible short delay during the operations;

4) the rails charged into the pits at a higher temperature are better aligned after cooling and, hence, their straightening is made easier. In addition, such rails are less liable to deformation during cold straightening and hence their mechanical properties are improved.

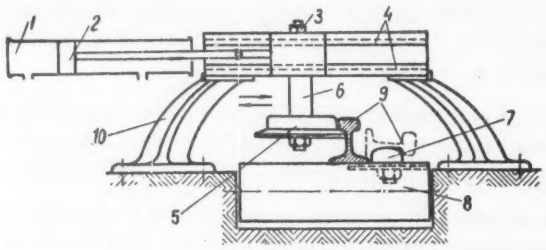

Fig. 2. Turning device for turning the rails through 90° after they leave the cambering machine. 1) Pneumatic cylinder; 2) piston and connecting rod; 3) slide bar; 4) parallel guides; 5) tilting disc; 6) disc shaft; 7) supporting roller; 8) roller of the roller tables; 9) rail before and after being turned through 90°; 10) stand.

The turning of rails behind the cambering machine. After leaving the cambering machine the rails are turned through 90°, "on edge," before being placed on the transfer gantry to be conveyed further by the transfer.

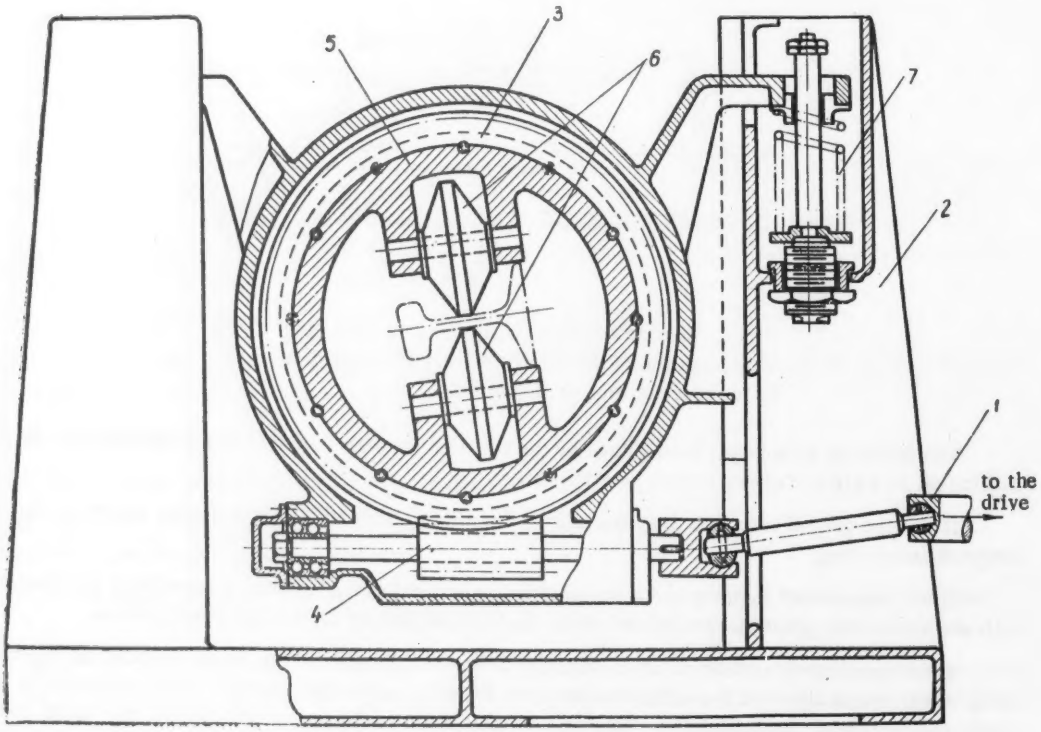

Fig. 3. Turning collar: 1) drive; 2) stand; 3) toothed rim; 4) worm gear; 5) frame; 6) gripping roller; 7) shock absorber.

The diagram of the turning device designed by I. A. Fabrikant, the engineer of the rail-structural mill, is shown in Fig. 2.

The rails are turned through 90° by means of the disc (5), which moves in the direction indicated by arrows, and the supporting roller (7). The shaft (6) of the disc is attached to the slide bar (3) which is moved on the parallel guides (4) by means of the piston and connecting rod (2). The stroke of the slide bar is 400-500 mm. The device is driven by compressed air in the cylinder (1). The disc and the supporting roller revolve freely under the action of friction between their surfaces and the surface of the moving rail.

The diagram shows the position of the disc, the supporting roller, and the rail being turned, at the initial stage of the turning operation; the base of the rail rests with the flange against the supporting roller, the disc is moved to the right and so turns the rail. Compressed air change is effected at the control desk by means of an electromagnetic valve.

The device proved to be suitable for turning any type of rail of various lengths.

The turning of rails on the gag press. The turning collar (Fig. 3), employed for turning rails on gag presses, is reliable in operation and fully satisfies the needs in this section.

The rail to be placed on the gag press is firmly held in the collar by the gripping roller. In the course of rail straightening, the collar is rotated by the electric motor through a worm gear. A shock absorber protects the turning equipment from shocks during rail straightening.

The turning collar is made suitable for turning rails of all types manufactured at the works.

The main advantages of this design are the compactness, ease of operation and maintenance, reliability in operation, and high throughput.

With a coordinated operation of the adjoining units (milling, drilling, and hardening machines) the gag press equipped with the turning collar shows a throughput of 450 or more 25-m-long R-50 rails per shift.

CENTRIFUGAL CASTING OF WATER PIPES IN SAND-LINED MOLDS

Z. A. Grinberg

Chief Engineer of the Pervouralsk Starotrubnoi Works

Of the known methods of centrifugal casting of small-diameter (50-80 mm) water pipes, the most efficient is the method of casting tubes in removable sand-lined molds. Until the present time, this method has been practically the only one employed in cast water-pipe production in the world.

Since 1956, work has been carried out at the Pervouralsk Starotrubnoi Works on the centrifugal casting of 2-inch diameter, 3.1-m-long tubes in green-sand molds. For this purpose, a special machine was made at the Works (Fig. 1).

The flasks enter the machine and then leave it through a special door at the side. In the machine, the flask rests on supporting rolls and is fastened with gripping rolls from above. It is then rotated at 800-1500 rpm, by a 15 kw motor, through a jaw clutch.

This casting machine is distinguished from previous designs by the device which makes it possible to incline the mold at 1.5° before it is filled.

The machine is moved from the horizontal to the inclined position by means of a wedge-shaped device,

actuated by an 11 kw electric motor. It takes 4 sec to change the position, the speed of the wedge being 125 mm/sec.

The position of the machine is changed before the mold is filled with metal but after the flask is inserted and fastened. After being filled, the machine is returned to its horizontal position and the tube is cast when the flask is in horizontal position. In this way any non-uniformity in the wall thickness along the pipe is prevented; this defect was inevitable on machines of earlier designs. The machine operates in any position without vibrations.

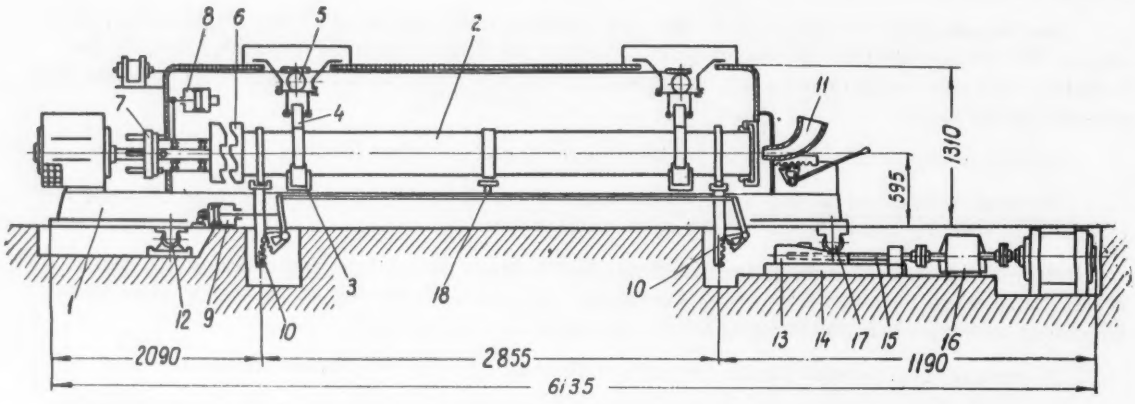

Fig. 1. Centrifugal casting machine: 1) frame; 2) flask; 3) supporting rolls; 4) gripping rolls; 5) fastening device; 6) jaw clutch; 7) coupling; 8, 9) air cylinders; 10) lifting device; 11) attachment for mold filling; 12) pivot axis; 13) wedge; 14) wedge guides; 15) screw; 16) gear box; 17) roller; 18) brake

To speed up fastening the flask, the one-turn screw of the fastening device was replaced by a two-turn screw, but eventually the pneumatic fastening was found to be most promising; the adoption of the pneumatic fastening resulted in an improved operation of the machine and the prevention of rapid wear of the flask hoops.

In order to cut down the time of flask rotation due to inertia (it constituted 1.2 minutes on the old-design machines) after the motor has been switched off, a pneumatic shoe brake is employed; and for this purpose the flask has an additional hoop.

The sand is rammed by means of the plunger-and-screw molding machine (Fig. 2) designed by Ia. M. Semenov, S. M. Bogorad and A. D. Drozdov.

From the distributing bin, the mixture for the mold is fed through the propositioning feeder and by the screw conveyor into the flask (fixed horizontally on the carriage of the machine) and is compacted by the last turn of the screw and by the plunger. The compression is controlled by a special shoe brake, with a sector adjustment, positioned on the carriage itself.

As the batches of molding material are fed and rammed, the carriage with the flask is gradually moving away until the whole flask is filled. The finished mold is delivered from the machine onto the gantry by pneumatic lifts.

A very dense, 27.5-mm-thick layer of molding material was obtained when the experimental molding machine was used. At the present time, tests on producing 18-mm-thick layers are being carried out.

Initially the entire pipe with the socket was formed, but later on the earth socket was replaced by a cast iron socket. During the operation of the molding machine, it was found that a good surface of the mold was obtained if the plunger was not rotating when the conveyor screw was turning. If the plunger rotated, helical impressions sometimes appeared on the mold surface. The plunger-and-screw molding machine ensures the production of good-quality molds for the manufacture of centrifugally-cast tubes and it has several advantages as compared with the Ardelt ramming machine and the pneumatic vibrating table.

A large amount of molding material is required when a vertical molding machine is employed, since, owing to the weak compacting and the particular design features of the ramming head, the thickness of the layer must not be less than 65-70 mm. The density of the layer formed on the horizontal machine is much higher and therefore it can be made 27.5 mm thick. Hence, it was possible to reduce the internal diameter of the flask to 120 mm (the diameter of the flask in use at the Makeevka Works is 195 mm). In this connection, consumption of molding mixture was reduced by a factor of about three.

The high density of the lining as obtained on the plunger-and-screw machine made it possible to dispense with the mold wash application and with the drying, necessary for obtaining strong skin in the case of other methods of molding production. If the lining is dense, the break-through of liquid metal through gas gaps in the flask does not occur.

In accordance with the accepted method, the tubes are cast in green-sand molds, the entire process thus being much simplified. The time required for the solidification of metal is then shorter, and the output of the machine is 20 tubes per hour. The thinner the layer of the sand mixture, the higher is the output.

A pneumatic-type apparatus is employed for the removal of sand from the flask after filling the mold and cooling.

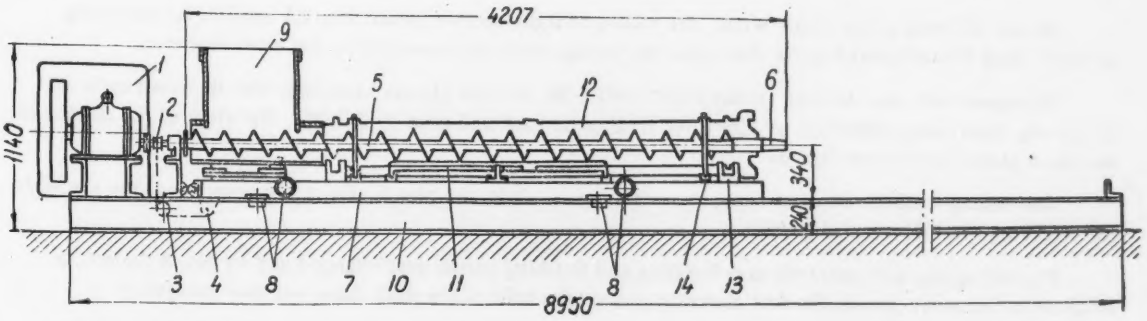

Fig. 2. Plunger-and-screw molding machine. 1) Main drive; 2) drum; 3) roller; 4) bracket; 5) screw; 6) plunger; 7) carriage; 8) supporting and guiding rollers; 9) feeder; 10) frame; 11) flask delivery device; 12) flask; 13) supporting diaphragms; 14) attachment for fastening the flask

The filled flask is transferred from the centrifugal casting machine to the stripping machine where the end plates are taken off and the flask is passed to the cleaning chamber.

The chamber is tightly sealed by swing doors covering the chamber openings.

Compressed air, entering the mold through a tubing, breaks up the burnt sand in the flask. The dust produced is removed from the chamber by exhaust equipment and is collected in a cyclone. Before being impregnated with asphalts, the tubes are cleaned of adhering sand on a special machine with revolving steel brushes.

The process described has one more advantage, namely that the flask is transferred only in the horizontal position from one unit to another, and hence the process can be fully mechanized and automated. Moreover, no high buildings, which are usually required for vertical molding machines, are necessary.

More than 4000 t of good tubes were cast on the experimental equipment consisting of one plunger-and-screw machine and two centrifugal machines.

The average weight of the tubes cast on the centrifugal machine was 1.7 kg less than rated weight, while the weight per meter length of tubes cast on vertical centrifugal machines by the combination casting method is always greater than the theoretical weight.

The production of such tubes was made possible by the fact that, on solidification in the mold, the casting is much sounder due to the action of centrifugal forces.

Investigations carried out on the reduction of the tube wall thickness provide a basis for the assumption that it will be possible to revise GOST 5525-50 relating to cast-iron water pipes and to provide for a 15-20% reduction in wall thickness and an increase in the length of two-inch pipe by a factor of 1.5, a substantial saving in metal thus being achieved.

At present, work is being continued on the improvement of the process of centrifugal pipe casting in green-sand molds.

ROLLING OF LIGHT-SECTION GIRDERS

I. N. Filipov and I. V. Gunin

Ukrainian Institute of Metals

On the 350 mill of the Kirov Works, the rolling of light-section girders No. 6.5 and No. 8, conforming to GOST 8240-56 and providing for more than 3% savings in metal consumption, has been mastered.

Compared with the old-type girders GOST 10017-39, the new girders have their web thickness reduced by 6.3-4%, their flange thickness by 1.5-7.5%, their cross-sectional area by 3-3.2%; the width of the flanges of the No. 8 girder is increased by 5%.

The rolling mill has 11 open-type, two-high stands. Only one pass is allowed in each stand, the last eight passes having specially shaped grooves.

For rolling No. 6.5 girder the pre-finishing and finishing passes were changed, and for No. 8 girders, in addition to the above passes, the first checking pass on the rolls of the sixth stand was also changed.*

The curvature of the girder over the height of the web amounted to 0.4 mm, and an underfilling of the web corners was observed. In one campaign, the mean weight per m length was 6.5 kg, i.e., equal to the theoretically calculated weight.

In order to eliminate the curvature and the underfilling of corners, the false flanges in the pre-finishing pass were slightly increased. During subsequent rollings, these defects did not appear.

The No. 8 light-section girder obtained from experimental rolling was also satisfactory, but one flange was not constant in width which decreased to 42.8 mm (nominal width = 45 mm, tolerance = $\pm$ 1.5 mm). The following modifications of the pass design were introduced in order to eliminate this defect: 1) the flange groove in the bottom rolls of the first and second controlling passes was made deeper by 2 mm and 0.5 mm respectively; 2) the height of the rectangular billet entering the first shaped pass was increased by 5 mm. These modifications resulted in a fully satisfactory and constant section produced in subsequent rolling operations.

The yield of acceptable product is roughly the same for ordinary and light-section girders. The output of Ist grade product constitutes 97-99%.

The weight per m length of No. 6.5 and No. 8 girders is 3.0% and 3.2% respectively less than the weight of the ordinary girder. The output per hour of light-section girders is theoretically reduced by only 1.5% and 1.6% compared with the output of ordinary girders and corresponding numbers.

After the rolling of light-section girders was introduced, the actual output remained as before (it decreased slightly for the No. 8 girder and increased for the No. 6.5 girder).

The idlings of the mill during the rolling of the light-section girder increased by 15-20 min owing to more frequent roll changes. The service life of grooves for rolling light-section girders is 1.3-1.5 times less

*Roll design by A. I. Solov'ev.

than for ordinary girders. Thus, for instance, the service life of finishing stand passes was 450 t and 400 t for ordinary girders No. 6.5 and No. 8 respectively, but it was only 300 t for light-section girders.

The main reason for the shorter service life of the passes is a lower temperature of rolled pieces at the end of the rolling process because of their smaller thickness.

FROM THE HISTORY OF TECHNOLOGY

BEGINNINGS AND INITIAL DEVELOPMENT STAGES OF THE BLAST-FURNACE INDUSTRY

Pig iron is usually regarded as the product of the blast-furnace industry but this view is not quite right. Chinese literature indicates that pig iron was known in China several centuries B.C. and it was not produced in blast furnaces but in crucibles.

The crucibles, about 400 mm high and 150 mm in diameter at the top part, made of refractory clay, were filled with a mixture of well-ground anthracite and ore, and were embedded in anthracite coal in a special hearth. Air was blown into the hearth, and the combustion of the anthracite provided heat to keep the crucibles and the material inside them red hot for about two days. As a result of this process globules of iron were formed in the crucibles. This method of iron smelting was not known to Europeans.

The advancements of the Chinese in the field of foundry work are indicated by a large cast iron container in which the code of laws was kept. This container, found in Shansi province, is thought to have been made in 513 B.C.

It can be inferred from the works of Aristotle (4th century B.C.) that he knew about pig iron. However, there is no description of any method of iron production in his writings.

In the literature of a much later period, the incidental production of pig iron in the furnace for wrought iron production – bloomeries and high bloomery furnaces (Fig. 1) – is mentioned. Since, however, in this case the yield of the bloom was reduced, the production of pig iron was considered to be undesirable. Later on, when metallurgists found that liquid pig iron could be poured into molds and could be used for the production of castings, the intentional production of pig iron was started.

Fig. 1. Work at high bloomery furnace (from Agricola's book)

The development of pig iron production began when it was established that making bloomery iron from pig iron is more economical than the direct production of bloomery iron from ore. The first documentary note on pig iron is in the works of Bishop Henrich of Poitiers written in 1371.

In the opinion of the German metallurgist Karsten the first design feature which distinguished the blast furnace from the bloomery furnace was the construction of the open-front hearth (Fig. 2). This construction was presumably necessitated by a high viscosity of slag obtained in the early blast furnaces.

The second design feature distinguishing the blast furnaces from the bloomery was a smaller hearth width which was probably aimed at increasing the temperature in the hearth.

The description of the construction and operation of a blast furnace was published for the first time in Filaret's novel "Architecture" in 1450. It is interesting that very soon (1460) irregularities in the operation of the blast furnace were mentioned in ecclesiastical literature. Of course, it was not the technical aspect of the problem that was discussed but a "miracle" performed by the bishop of Florence, Antonius, whose prayers helped in removing the "skull" formed in the furnace.

The height of the first blast furnaces was not substantially different from the height of the bloomery furnaces, and according to B. Newman was 4-5.5 m. Externally, the furnaces constituted a massive rectangular tower whose outside wall was built of ordinary non-heat-resistant stone reinforced by iron or wood; the internal part of the furnace was built of natural heat-resistant stone.

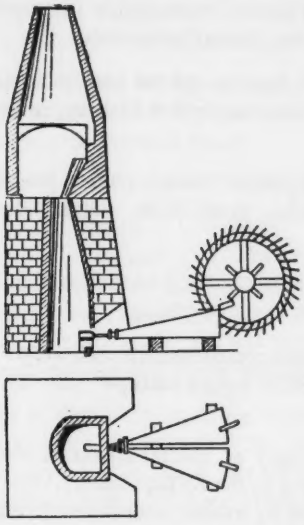

Fig. 2. Open-front shaft furnace for pig iron production. Air was blown with bellows driven by a water wheel.

Air was blown into the furnace by means of bellows (Fig. 3), actuated by a water wheel. Usually a furnace was provided with two bellows. When one was blowing the air the other was taking the air in so that a uniform blast was produced in the furnace.

Charcoal was used for fuel; most diverse types of iron ores were used. Small deposits of iron ore were found adequate for the limited production of pig iron.

Only fragmentary information can be found regarding the output of old blast furnaces. Vedding reports that in 1544, the blast furnace in Ilsenburg, Harz, was producing 720 kg pig iron per 24 hrs; according to Svedenborg the output of a bloomery furnace in the beginning of the XVIIth century was 200 kg per 24 hrs. According to Johannsen, the output of the blast furnaces from the XVIth to the XVIIth century varied within the limits of 0.8 t-1.6 t per 24 hrs, and the consumption of charcoal was approximately two unit weights of charcoal per unit weight of pig iron.

The first reliable information on blast furnaces in Russia dates from 1632, when the first works were built on the river Tulitsa, near Tula. Several other works with blast furnaces were soon built in Central Russia. A rapid development of the blast furnace industry in Russia began only in the XVIIIth century when the metallurgical works in the Ural were put into operation.

Fig. 3. Blowing air into the blast furnace by means of bellows.

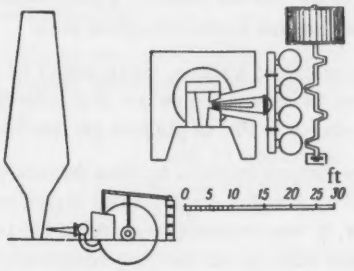

Fig. 4. Large blast furnace at the Nev'eansk Works

The blast furnace industry in the USA began much later than in West Europe. In 1619, the Virginia Commission began the construction of iron works at Falling Green, but in 1622, Indians killed the workers and demolished the works. The second iron works was built near Boston and its blast furnace was put into operation 1645-1647. Very few new works were built in the XVIIth and XVIIIth centuries, since North America was then an English colony

and England hindered the development of the American iron industry in various ways; in 1750, the English parliament passed a bill which prohibited the development of the metallurgical industry in America.

The development of blast furnace production and of the design of the blast furnace was very slow from the XVth to the XVIIth centuries. The height and the width of the blast furnace were slightly increased, air-blowing means were improved, and the design of auxiliary equipment was gradually improved.

In the beginning of the XVIIIth century the height of the largest furnaces did not exceed 10 m. According to Newman, the blast furnaces in France, Belgium and Germany were usually 7 m high and in England they were 8-8.5 m high.

In 1740, at the Nev'iansk Works of Prokop Demidov, a 12.8-m-high blast furnace (Fig. 4) was built; its output, which was regarded as extremely high, approached 14 t per 24 hrs. In the Urals, ordinary blast furnaces, operated with charcoal, produced about 4-4.5 t per 24 hr.

According to L. Beck and B. Newman, the output of blast furnaces in Germany varied between 0.7 to 3.3 t per 24 hrs, in England – from 1.0 to 3.0 t, and in Sweden – about 0.6 to 3.0 per 24 hrs.

According to German, the consumption of fuel at the Ural furnaces constituted 1.2-1.65 weight units per weight unit of pig iron, in Siegerland it was 1.6, and in Styria 2.5-2.75 weight units per unit of pig iron.

The increase in the height of the blast furnace entailed the necessity of special means for charge delivery to the furnace top. Hoists, driven by a water wheel, were invented in the XVIIth century. The capacity of the air-blowing means was increased. Leather bellows were replaced by wooden ones (about 1550) and these in turn were replaced by box-type air blowers and later on by cylindrical air blowers (at the end of the XVI century). The output of leather bellows was somewhat more tha 1 cu m/min, of wooden bellows – about 8 cu m/min and of box-type air blowers – about 12 cu m/min.

Thus, there was no substantial change in the blast furnace industry from the XVth to the XVIIIth centuries. As before, water served as the main driving power and charcoal was used as fuel.

The lack of fuel in some countries hindered the growth of pig iron production. Thus, owing to the shortage of charcoal, only 59 blast furnaces were in operation in England in 1740 and pig iron output was only 17,350 t; whereas in the XVIIth century, according to information by Dodley, there were 300 furnaces in operation and pig iron output amounted to 180,000 t in 1613.

The change-over from charcoal to mineral coal took place for the first time in England in the XVIIIth century. Without dwelling on the subject, since it has been already discussed,* we shall only mention that initially the application of coke instead of charcoal in the blast furnace gave rise to a number of technological difficulties and did not result in a noticeable increase in pig iron output in spite of the obvious advantges of coke over charcoal as metallurgical fuel.

According to Karsten, mean output of a West European charcoal-fired blast furnace in 1788 constituted 2-2.5 t per 24 hrs and the output of a coke-fired furnace constituted 3.0-3.2 t. Siemens writes that Abraham Derby produced 5-10 t of pig iron per week at his works at Cobrookdale.

The limited increase in blast furnace output in the inital period of coke introduction can presumably be explained by a higher density and a higher resistance of the stock to the passage of gases. To overcome this resistance, it was necessary to increase the pressure and the amount of air blown into the furnace, but this could not be done with the air-blowing equipment driven by a water wheel. Only when steam-driven air-blowing machines were introduced did the operating efficiency of coke-fired blast furnaces begin to increase markedly.

According to one source, the first cylindrical air blower driven by steam was introduced in England at the Bradley Works in1775, and according to another source it was introduced first at the Carron Works in Scotland.

In 1763, an eminent Russian engineer, I. I. Polzunov, developed a project for the application of steam for driving the air blower. The machine began to work in 1766, after the death of the inventor. It was not,

* "Metallurgist", No. 5, 1937

however, the steam engine that was adopted in this equipment but the so called "fire-acting" machine of Newcomen, improved by I. I. Polzunov; that is to say a machine in which the piston was actuated by atmospheric pressure after a vacuum was established on the other side of the piston as a result of steam condensation caused by injected water. The principle of operation of this machine was different from the principle of the Watt engine, where the piston was moved as the result of steam expansion.

At the end of the XVIIIth century, Russia occupied the first place in the world in the production of pig iron owing mainly to the Urals iron industry. Some data on the production of pig iron in Russia and in England are given below (in thousand tons):

	1700	1740	1760	1780	1800
Russia	2.5	25.1	60.0	110.1	172.4
England	12	17.4	27.0	40.0	156.0

In other large industrial countries the annual output of pig iron at that time was considerably lower. In France, only 69,100 t of pig iron was produced in 1790 and 140,000 t in 1800. Annual output of pig iron in Sweden, in the second half of the XVIIIth century, was maintained at the level of 60,000 to 65,000 t.

In the XVIIIth century, Russia occupied the first place in the world not only in the output of pig iron but also in the technical level of the blast furnace industry. In 1743, Grigorii Makhotin commenced air blowing through two tuyeres (Fig. 5). It was introduced much later in Western Europe. The output of the blast furnaces in the Urals was usually 5-6 t per 24 hrs, but often it approached 10 or more per 24 hrs.

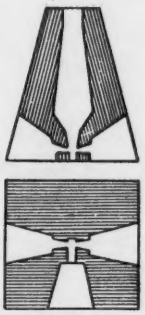

Fig. 5. Two-tuyere blast furnace of the Nev'iansk Works

Large quantities of iron from the Urals works were exported to England. Thus, 1000 poods (16 t) of iron were exported to England in 1716, 9600 t in 1749, 13,100 t in 1758 – 1760, 32,800 t in 1778 – 1780, and 43,000 t in 1790 – 1792. In the beginning of the XIXth century, the export of metal to England began to decline.

The technical decline in the metallurgical industry in Great Britain was caused by a rapid reduction in production, owing to the shortage of charcoal. Only after the introduction of coke and the cylinder air blower did Great Britain rapidly develop its metallurgical industry and left all the main countries of the world far behind in the field of ferrous metallurgy.

From the beginning of the XIXth century, Russia gradually lost her lead in the blast furnace industry and by the middle of the century she was behind many West European countries.

Only after the victory of socialism was our country able to make up her long-standing lag and to take the leading place in metallurgical and other branches of industry, leaving Great Britain and many other European countries far behind.

Cand. of Tech. Sci.
N. I. Krasavtsev

NEW BOOKS

M. L. MIRENSKII, WORK AT A LIGHT SECTION MILL

Moscow, Metallurgy Press, 1957, 263 p

This book contains short general accounts on the classification, properties,and teeming of steel, on ingot defects and on plastic forming.

The theoretical principles of longitudinal rolling and the characteristics of the deformation are treated in adequate detail; there is information on the spread and grip of metal, slip during the rolling pressure of the rolled metal on the rolls,and the work of deformation.

The classification and operating statistics of rolling mills are given, and the main rolling equipment, roll housing,and auxiliary equipment of rolling mills are described. Special attention is paid to the operation of automatic-transfer equipment.

The chapter on roll design is devoted to a description of the main methods of rolling design, the construction of the grooves, and the arrangement of the passes for rolling simple sections.

A shortcoming of this chapter is the lack of design calculations of passes for special,shaped sections rolled on light-section mills.

The short description of the technology of the section-rolling process elucidates the principles of heat-treatment and rolling of light-section steel; the equipment of the modern, mechanized,and highly efficient mills and their layout, as well as the adjustment of the mills, are described.

The unfortunate example of the layout of an old-type light-section mill (page 228) should have been replaced or supplemented by a more modern one – a three-line mill provided with coolers.

Unfortunately, this book for foremen contains no information on such production data as operating efficiency on rolling various sections, the definition of the work coefficient, metal consumption, production cost, etc.

The book is written quite lucidly; it is well arranged and can be used as a textbook for foremen and mill operators of light-section mills.

METALLURGY ABROAD

TUBE PRODUCTION IN ENGLAND

Iu. M. Matveev

Gipromez

At present, England occupies the second place in Europe in tube production (after the USSR) and is producing about 1.5 million tons of tubes annually. Before the Second World War (in 1938) tube output in England was 676,000 tons, in 1956 – 1,447,000 tons and in 1957 (according to preliminary estimates) – 1,500,000 tons. Half of the tube output consists of welded tubes.

The production of tubes in England is concentrated at the works of Stewarts & Lloyds and at Tube Investment. Stewarts & Lloyds produces over 1 million tons of tubes annually, i.e., about 70% of the total tube output in England. Stewarts & Lloyds has its works in Corby, Glasgow, Birmingham, and Cardiff. The works of the Tube Investment Company produce various special tubes including tubes made of alloyed and highly alloyed steels.

The most interesting of the works of the Stewarts & Lloyds Company is the Corby Works, comprising tube-welding, electric tube welding and tube rolling plants. The total output of tubes at this works exceeds 600,000 tons per year.

In the tube-welding plant there are four continuous mills for the manufacture of $\frac{1}{4}$ to 1 in diameter gas pipes (see table). In 1954-1956, the butt-welding tube plant was modernized. The thermal capacity of the welding furnaces was increased; they were partly converted to oil heating; the drafts in the roughing stands were increased; additional stands were built and hot saws with revolving carriage were installed.

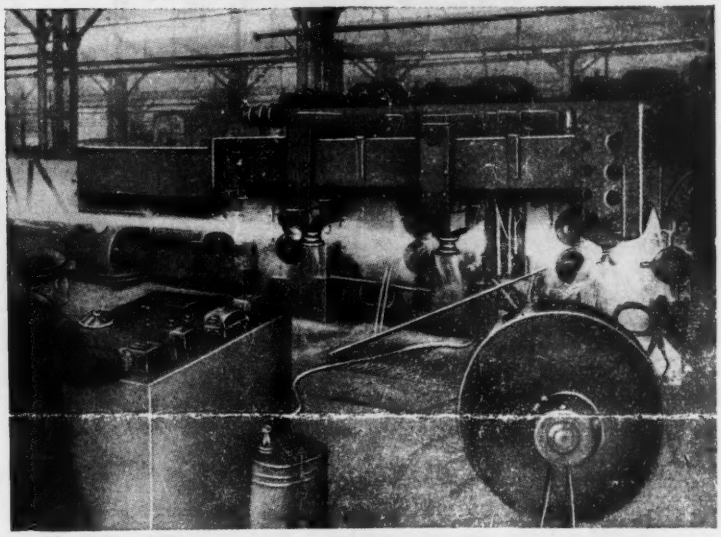

Fig. 1. Continuous butt-welding tube mill for $1\frac{1}{4}$ to 4 inch diameter tubes

The burners (jets) are distributed over the whole length of the heating furnace, including the zone of preheating where they were not employed previously. In connection with the increased number of burners, additional recuperators were introduced. One third of the furnace from the end of the strip exit is fired with gas, and in the remaining part of the furnace, burners for fuel oil are installed. The skelp passes above the roof of the furnace over about one third of the length of the furnace and at that point it enters the furnace.

Main Characteristics of Continuous Furnace-Welding Tube Mills at the Corby Works

Diameter of manufact. tubes in inches	Length of loop, m	Type of loop	Number of stands of the butt welding mill	Type of gear of the butt-welding mill	Saw type	Maximum exit velocity of the tube from the mill m/min	Year established	Year modernized
1/4—3/4	35.0	Single	6	Groupped	Crankshaft-tiller	140	1934	—
3/4—1	49.2	same	8	same	Revolving carriage	160	1934	1954
1—2	49.2	" "	8	" "	same	220	1944	1956
1 1/4—4	55.0	Double	8	Individual	Crankshaft-tiller	180	1947	—

The skelp then passes over the upper row of gliding tubes to the charge end of the furnace (during that time it is heated to 350-400° C), turns around the driving roll and passes, on the lower row of gliding tubes, to the welding machine. This arrangement of skelp movement (it passes twice through one third of the furnace at the charging end) improves the utilization of the furnace and reduces fuel consumption.

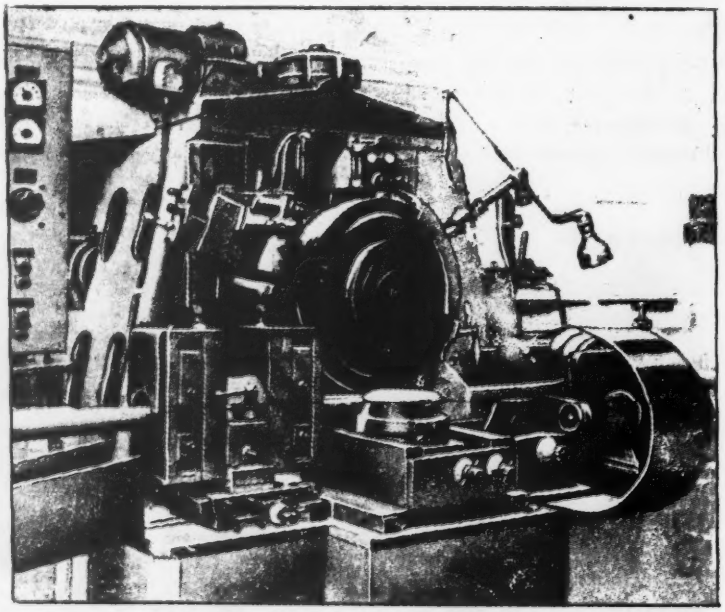

Fig. 2. Electric tube welding machine.

After the modernization, the width of the skelp was increased by 20-25%, owing to the increase in the draft at the reducing stands and to the introduction of two additional reducing stands (Fig. 1). The exit speed of tubes increased to 220-250 m/min. For cutting tubes as they are moving, saws, on a carriage revolving in a vertical plane were installed, the tubes being cut into 4 m-16 m lengths with an accuracy of ±10 mm.

After cutting and straightening, the tubes are transferred to the intermediate storage and then go to one of the seven finishing trains. Each finishing train consists of two tube threading machines, socket drilling machine and a hydraulic test press. In each unit, two tubes are worked simultaneously. The tubes are painted by the electrostatic method.

The plant for electro-welded tube specializes in the production of boiler tubes of 25-114 mm diameter. There are two machines, 25-63 mm and 51-102 mm respectively, for the electric-resistance welding of tubes (Fig. 2). Electric current of 250-350 cps is fed to the welding transformers in order to improve the quality of the seam. The tubes are welded at a speed of 35-40 m/min. Only the most commonly used sizes are welded. For the production of intermediate sizes, there is a fully automated drawing mill of 40 t pulling force (at a simultaneous stretching of two tubes) designed for 60 m/min drawing velocity.

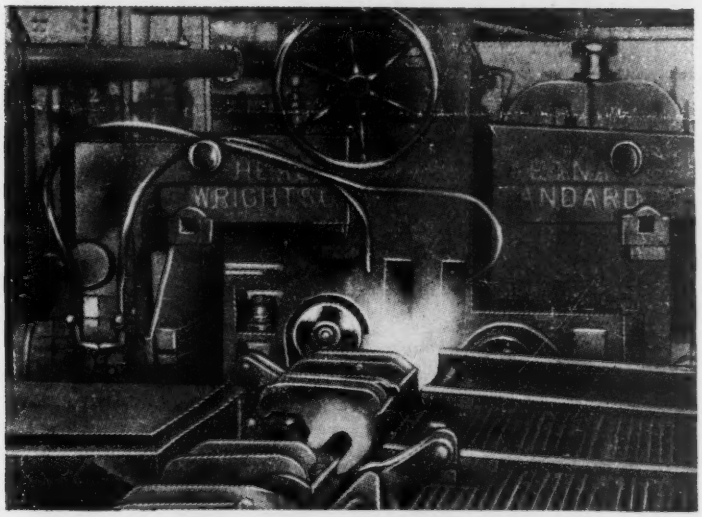

Fig. 3. Piercing mill.

The thermal treatment of tubes is carried out in three roller furnaces with controlled atmosphere. The furnaces, including the cooling chamber, are 55 m long. There are two automatic tube-finishing trains each consisting of a straightening machine, two cutting machines, two sizing machines, and a hydraulic test press.

In 1954, the automated mill for rolling 50-140 mm diameter tubes was installed at the tube rolling plant. The plant comprises: a furnace with a circular rotary hearth, a piercing mill (Fig. 3) driven by a synchronous motor, a heating furnace, an automatic mill (Fig. 4), two reeling machines, and a seven-stand sizing mill.

Among the remaining tube works in England, the Clydesdale Works, where a pilger mill for rolling tubes of 114-245 mm diameter was installed in 1956, is of interest. The initial material for the tubes is twelve-sided ingots whose diagonal is twice the diameter of the finished tube. Ingots are heated in the three-zone holding furnace, with the hearth consisting of separate buggies. The heated ingot is transferred to the hydraulic press where it is end-compressed to remove the scale. In the piercing machine, the ingot is fixed in the holder and is pierced almost to the end. The "cup" produced is transferred to the piercing mill. The pierced shell is heated in the holding furnace and transferred to the stretch mill and then to one of the two stands of the pilger mill. After the sizing operation at the seven-stand sizing mill, the tubes are subjected to finishing operations. This section is fully mechanized and automated.

In England seamless tubes are produced mainly on pilger mills, most of which (Clydesdale Works, Bromford Works and others) have piercing presses. Pilger mills of the old design are also in operation (London Works).

Only tubes of less than 140 mm diameter are produced on automatic mills (Corby Works and Tallcross Works. At one of the works (Combswood), a plug rolling mill, recently completely modernized, is in operation.

In 1956, the South Durham Iron and Steel Company put into operation at Stilton-on-Tees a plant for the production of large diameter welded tubes by electric arc welding with a layer of flux. The billets at this plant are prepared on presses. At the British Works af Stewarts & Lloyds, large diameter tubes are welded by means of water gas.

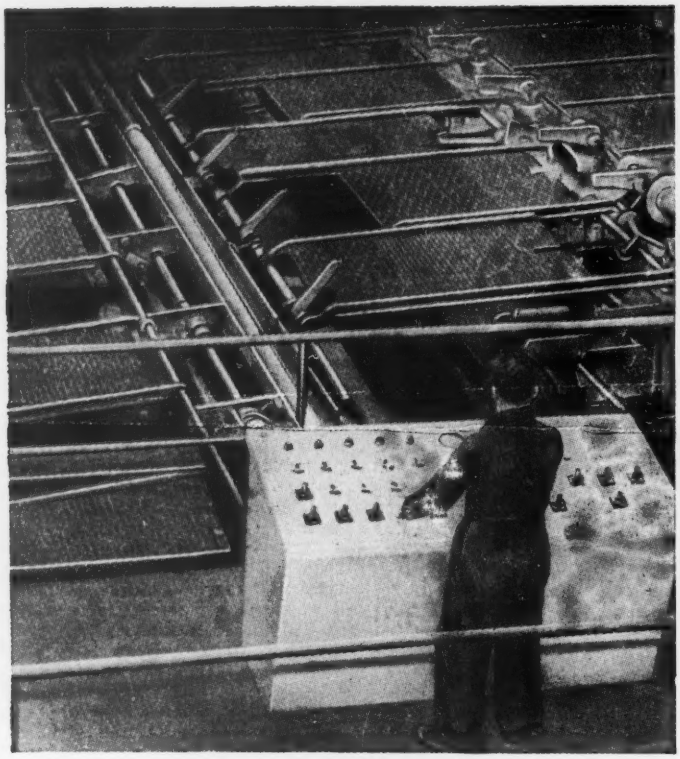

Fig. 4. Automatic mill.

In addition to the Corby Works described above, electric resistance welded tubes are manufactured also at a few works of the Tube Investment Company near Birmingham.

On July 20, the Soviet people celebrated Metallurgist's Day. Soviet metallurgists completed the output plan in all branches of iron and steel production for the first half of 1958 ahead of time. Thousands of tons of iron ore, pig iron, steel,and rolled products over and above the plan were produced.

180 workers of the ferrous industry who achieved outstanding production successes were awarded the title of a Hero of the Socialist Labor and about five thousand people were awarded Soviet orders and medals.

The great plans for a further vigorous development of our national economy call our metallurgists to new, still more glorious industrial achievements.

Inspired by the care and attention of the Communist Party, of the Soviet Government,and of the whole nation, the workers of the ferrous industry will develop, on an even more extensive scale, the socialist competition for the fulfillment of the production plan for 1958 in all sections of the industry, ahead of time.

We heartily congratulate those on whom orders were conferred and the Heroes of Socialist Labor!

We wish Soviet metallurgists new successes in their work for the good of our beloved Fatherland.

Editor

NOTE

A Questionnaire Asking for Comments and Suggestions by Readers of Metallurgy, Included with this Issue, has not been Included in the Translation – Publisher.

Patriotic initiative of miners from Donets, Nikolai Mamai, and Aleksandr Kolchik found a response among Soviet metallurgists. At the open-hearth plant of the "Azovstal" Works, the team of steel melters led by Nikolai Pereverzev were the initiators of a movement for the elimination of defective production and an increase in the output of first-grade product. The initiative of this team has been taken up by several metallurgical works of the South.

The picture shows Nikolai Pereverzev's team. From left to right: G. Ia. Gorban', V. V. Moiseenko, A. I. Maslianskii, N. V. Pereverzev.

Photo M. Medukhovskii.

SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosenergoizdat	State Power Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LEIIZhT	Leningrad Power Inst. of Railroad Engineering
LET	Leningrad Elec. Engr. School
LETI	Leningrad Electrotechnical Inst.
LETIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MEP	Ministry of Electrical Industry
MES	Ministry of Electrical Power Plants
MESEP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTI	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NII ZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOE	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIEL	Central Scientific Research Elec. Engr. Lab.
TsNIEL-MES	Central Scientific Research Elec. Engr. Lab.–Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIESKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZEI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us. – Publisher.

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